# Feasibility Study of a Continuous Borer-Bolter in an Underground Potash Mine

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## Abstract

Potash mining in Saskatchewan is most commonly done using one of two underground dry mining techniques: A variation of room and pillar mining, and stress relief mining. The potash ore that is mined has a large areal extent, is located at great depths, and is typically flat lying. However, some pockets in the ore body vary from the general geology, and are called anomalous. These areas can result in reduced stability from what would normally be found in the typical geological conditions. Potash mines install rock bolts in anomalous ground conditions to counter this. This is currently done using two different machines and results in a lot of down time for the mining operation. It has been theorized that by combining the two machines into one, time can be saved from the current process and result in increased stability of the mine back.

This study pertains to the feasibility of attaching a bolting system to the back of a continuous boring machine for excavating anomalous ground conditions in underground potash mines. Three parts of this are examined: an estimate of the time saved by changing bolting methods, the change in mine roof stability due to the different bolting method, and the feasibility of crafting a machine that can fit in the confined space required. Bolting procedures are examined using individual task rates and projected for the proposed bolting methods. The change in stability that results from changing the bolting procedure is analyzed using finite element analysis. Finally, a concept machine is devised to show the ability to put such a bolting machine in the confined spaced available, and therefore showing the possibility of implementation.

The geometric constraints, determined by a theoretical system, were used to create several feasible processes for installing bolts from the boring machine. This resulted in four potential variations in the process with regards to both the potential saved time and the stability of the tunnel. It was found that the potential for time savings were very large, with up to 72% of the current bolt process time being saved in some situations. It was found that the largest part of the mining process in poor ground conditions is spent on the installation of bolts, and therefore the most time was saved by methods capable of bolt installation while the boring machine was in motion. Based on modelled stress metrics in areas requiring support, the current support measures appear more effective than the proposed bolting procedure. The analyzed stress metrics were still improved when compared to an unbolted ground scenario; Similar values of simulated curvature and displacement were found in the tunnel back at lower disseminated clay heights when partially bolted than were observed when the simulation was unbolted. This study shows the borer mounted bolter can be considered an advantageous mining method for efficiency purposes in poor ground conditions and could potentially be suitable in some geological conditions, but likely should not be used in all anomalous ground scenarios.

## Acknowledgements

I would like to thank the professors who have helped me to reach this point in my degree program. My supervisor, Travis Wiens, who provided me with advice, helped me when I was unsure of where to go next, and kept me on track to finish this study in time. To professor JD Johnston for his help in guiding me through the requirements of the study. And to Paul Hughes who worked with me on understanding finite element analysis in geo-mechanics and helped me determine inaccuracies in the results.

I would also like to thank the companies who supplied the software used in this study. Educational licenses supplied by Autodesk for Inventor®, and AutoCAD®, as well as RocScience licenses for RS$^2$® and RS$^3$® finite element modelling programs helped greatly in the completion of this study.

Finally, I would like to thank Potash Corporation of Saskatchewan Inc. for funding and working with me on this study. They were very willing to let data be collected for the purposes of the study. Study funding was also provided by PotashCorp. Janelle Appleyard, Craig Funk, Owen Gunther, and numerous others gave up their time to help facilitate this study and get it finished in a timely fashion.

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## Nomenclature

| | |
|---|---|
| FEA | Finite Element Analysis |
| $a$ | the mining advance rate |
| $b$ | the movement rate of borer when not mining |
| $c$ | the time delay to move sideways from first to second pass |
| $d$ | the distance between rows of bolts in the direction of advancement |
| $e$ | the individual bolt installation time |
| $f$ | the time required to change machines |
| $g$ | the time required to change borer bits |
| $j$ | the number of bolt rows reachable by the bolter at once |
| $k$ | the time taken to move bolter from one bolting position to another |
| $l$ | the length of the cutting cycle |
| $m$ | the bolter movement rate |
| $n_{1st}$ | the number of bolt install cycles per first pass pause |
| $n_{2nd}$ | the number of bolt install cycles per pause during second pass |
| $n_{bolts}$ | the number of bolts required to finish the width of the entry |
| $n_{cut}$ | the number of cutting cycles required for a completed entry |
| $n_{pass}$ | the number of passes in the finished entry |
| $o$ | the brattice hang rate |
| $p$ | the distance mined before changing bits |
| $q$ | the extra distance left to allow free movement of the bolting machine |
| $r$ | the brattice removal rate |
| $t$ | the total time used by the anomalous ground cut and bolt procedure |
| $t_{mining}$ | the time spent mining |
| $t_{bolting}$ | the time spent installing bolts |

| | |
|---|---|
| $t_{BorerMove}$ | the time spent moving the borer without mining |
| $t_{BolterMove}$ | the time spent moving the bolting machine |
| $t_{Brat}$ | the time spent handling brattice |
| $s$ | the drilling cycle time |
| $u$ | the time to change the drill bit |
| $v$ | the number of bolts between changing drill bits |

# Chapter 1 Introduction

In Saskatchewan, the economy is driven mainly by natural resource production. One heavily produced resource is potash with several large mines operating in the province in various locations. According to Natural Resources Canada, approximately 46% of the global reserves of this mineral are found in Canada, of which a majority is found in southern Saskatchewan in the Prairie Evaporite Formation [1]. It is important that this industry can compete in the world market while still ensuring the highest levels of safety.

There are several geological layers and conditions that affect the ability of miners to effectively operate in the potash bearing layers of Saskatchewan. The three formations that primarily affect mining conditions are the Dawson Bay, Prairie Evaporite, and Winnipegosis formations. The Prairie Evaporite Formation is the potash-bearing formation in Saskatchewan. Within the Prairie Evaporite Fromation, there are four members that contain mineable concentrations of potash with varying ground conditions: Patience Lake, Belle Plaine, White Bear, and Esterhazy members [2]. The ground conditions may be further influenced by the structure of the Dawson Bay Formation above, and Winnipegosis Formation below [3] [4]. Each of the three formations affect potash mining in substantive ways.

One important condition is the depths the deposits lie at. In the Saskatoon area, the northernmost area potash is mined, the deposits are found at depths of approximately one kilometer with the depth increasing towards the south [5]. In the northern United States, the deposits can be found at depths of 3.5 kilometers [6]. The depth can be problematic as potash is a soft rock material that readily deforms at very low stress levels. The inherent stress in the material at one kilometer of depth, approximately 25 to 27 MPa, is very close to the unconfined compressive strength of potash [7]. Any excavation under these conditions will create deviatoric stresses high enough to causing slow closing of mined openings or potentially dangerous rock falls [8]. The depth of potash deposits can dictate how they are mined.

To deal with the inherent stresses and extent of clay seams in the mining horizon, several mining methods have been developed. The three methods used are long room and pillar mining, stress relief mining, and solution mining [4]. Currently, most mines operating within the province use one of the first two techniques, long room and pillar and stress relief mining [9]. In the eastern areas of the province, where ground conditions are generally much better, the method of choice is long room and pillar mining [9]. This method is simple to plan and has very few corners for boring machines to go around. In the Saskatoon area, the stress relief method is usually employed [10]. This method can be used in areas with extensive clay seams and at greater depths than are otherwise attainable with long room and pillar mining. Both methods result in approximately 35% to 40% extraction ratios, with some individual panels approaching 50% [4]. Solution mining, an occasionally used technique, especially in areas with extreme potash depths, requires no personnel or equipment to be in the mine workings underground. Each method is used in specific scenarios where the ground conditions allow.

While mining patterns are used to deal with long term stress effects, other methods are used to ensure safety for the mining personnel in the shorter term. Roof bolting is the method used in the Saskatchewan potash mines in anomalous, or otherwise poor, ground conditions. Roof bolting is used to serve two purposes: to pin multiple bedded sedimentary rock layers together, and to prevent large slabs of potash from falling out of the back (the "back" is also known as the mine excavation roof). Rock layers are pinned together to provide a roof beam with a much greater effective height, and in turn, greater structural integrity against buckling [11].

An image of the failure mode in layered rock environments is shown in Figure 1.1. In a potash mine, the clay seams, depicted in Figure 1.2, act as the low-strength bedding planes shown in Figure 1.1. Pinning these layers together across clay seams provides much increased levels of stability in the mine workings. Secondly, the bolts are used to support slabs of potash rock that could otherwise fall into the mine openings. This point is particularly important for mine safety. The installed bolts provide the most immediate back support if they are installed quickly after the original excavation, although installing too quickly can lead to the bolts prematurely failing because of excess ground movement. Similar yielding of the floor is common and referred to as floor heave. Rock bolts are very effective tools for ensuring short term safety in the mine workings.

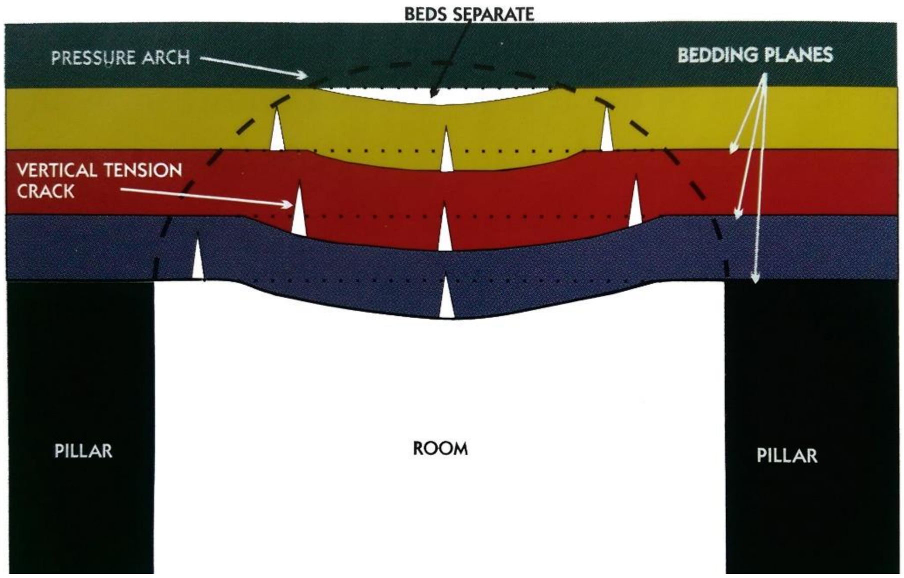



*Figure 1.1 - The general failure mode in layered rock environments. Saskatchewan potash mines the room can be between 9 and 18 meters wide. Tension cracks would only appear in the bent regions of the beams. Reproduced by permission of Workplace Safety North, [11].*

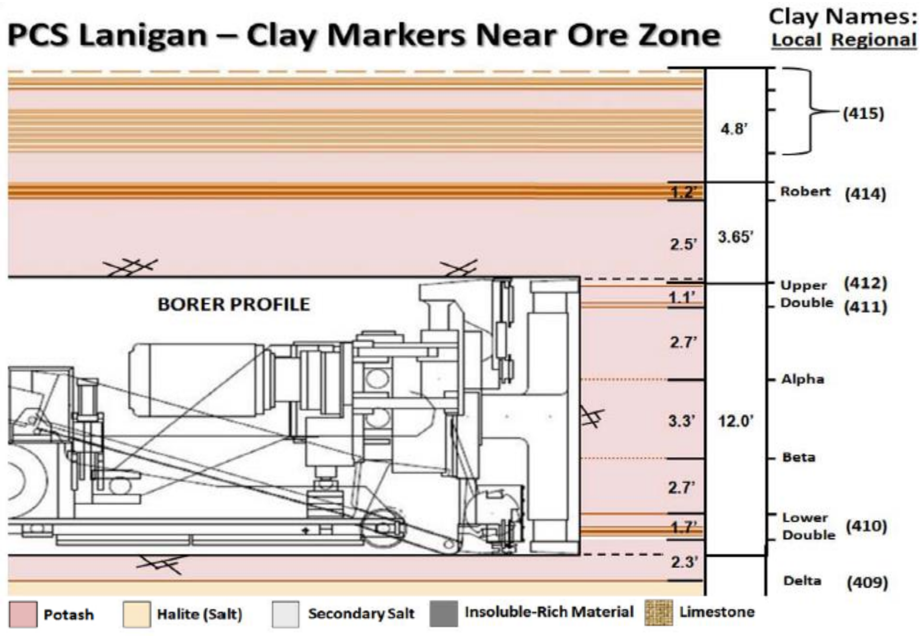



*Figure 1.2 - Diagram of the normal bedding sequence near the mining horizon at PCS Lanigan. The clay seams, such as 'Robert' (414), create planes of separation allowing the potash to fail more easily. Reproduced by permission of PotashCorp, [12].*

Rock bolts are used only in areas where they are deemed necessary in Saskatchewan potash mines. The current process for installing bolts while mining underground is a slow process, making mining slower than when bolts are not installed, so installing bolts where unnecessary adds to the operating costs of the mine. Although the potash ore is stressed to nearly its unconfined compressive strength even when undisturbed, it is known that the mines are still safe when mining activities cause the material to be stressed much more [8]. Bolts are generally only considered necessary in anomalous ground conditions, where the sedimentary rock layers are altered from the normal flat and consistent layering sequence [12], and thinner salt beams that tend to be more brittle result [12].

Installing rock bolts in underground mines requires specialized machinery. There are currently two different machine configurations that can accomplish this task: the specialized bolting machine, and the miner-bolter configuration. Specialized bolting machines are used for this purpose in a wide variety of situations, including potash mining. These systems are very versatile in the styles of bolts that can be installed. Installing the bolts generally requires that mining be stopped in an area and potentially includes moving boring machines to allow greater access, something that can take considerable amounts of time or leave longer sections of ground unbolted. The miner-bolter configuration can improve on this system from both a productivity and safety standpoint. The miner-bolter system attaches bolting arms to a non-continuous boring machine chassis to allow one machine to both mine and install bolts. This method does not require maneuvering the boring machines to allow bolting, allowing bolts to be installed sooner and quicker. The non-continuous mining style of these machines is, however, not as productive in normal ground conditions as the continuous boring machines currently in use in Saskatchewan. An optimal combination of both speed and safety is not achieved using either configuration.

There may be considerable potential to improve both the productivity and safety of the potash mines in Saskatchewan through the design of a new bolting system. The miner-bolter systems currently available to mines are an improvement over currently used methods in anomalous or otherwise poor ground conditions but do not offer the same production capacities as continuous machines when cutting non-hazardous ground conditions. The boring machines currently used mine at their higher designed rates in good ground conditions but the production rate is reduced, greatly in some cases, when anomalous ground conditions are present. Combining the advantages of both systems into a single system could increase both the production and safety levels in potash mines.

## 1.1 Thesis Outline

In the following chapters, this thesis will explain in detail the work undertaken to determine the feasibility of mounting a bolting system on a currently used boring machine. There are several parts that make up the whole study and they are generally divided into unique chapters in that fashion:

- Ch 2. LITERATURE REVIEW: A literature review of material pertaining to the study. Much of the information regarding exact mining rates and dimensions is either not available or proprietary so the literature review focuses on the geological conditions, general potash mining methods, and ground support styles.
- Ch 3. RESEARCH GOALS: The goals and end deliverables for the study are explained.
- Ch 4. PRODUCTIVITY MODELLING OF CURRENT AND PROPOSED PRACTICE: The rates of individual tasks were used to determine how time is used in the current anomalous ground bolting procedure and compared to several proposed alternatives.
- Ch 5. TUNNEL STABILITY MODELLING: The change in stability has been determined via finite element analysis to determine the safety of altering the cut and bolt procedure.
- Ch 6. CONCEPT DESIGN: A brief explanation of the requirements and theorized design.
- Ch 7. RECOMMENDATIONS: Recommendations based specifically on the results of chapters four and five including limitations of the system and proposed procedures.
- Ch 8. CONCLUSIONS: A conclusion on the effectiveness of using the borer mounted bolting system behind a continuous boring machine.
- Ch 9. FUTURE WORK: Contains some potential further investigations based on the information developed in the thesis.

## 1.2 Site Introduction

The information in this thesis is developed specifically for the Potash Corporation of Saskatchewan Inc. Lanigan mine site near the town of Lanigan, Saskatchewan. While the potash ore deposits in Saskatchewan are generally consistent throughout the ore members they lie in, and therefore, the general methods and ideas will apply to other potash mine sites, not all values used will transfer exactly to other mine sites. The PotashCorp Lanigan Division mine is in the Patience

Lake geological member at a depth of one kilometer below ground level [13]. This member is generally flat laying with high insoluble content in the form of clays. The clays are in separate distinct seams within the ore body. Immediately above the mining horizon there is one large clay seam of between 0.3 and 0.4 meters in thickness. The Patience Lake member has been described in more detail in Chapter 3 of this thesis.

# Chapter 2 Literature Review

There are several background aspects that affect the study outlined in this thesis. These include: the geology of the potash bearing members in Saskatchewan; the potash mining methods used; and finally, the ground support materials and machinery currently available.

## 2.1 Saskatchewan Potash Geology

The ability to mine the potash reserves in Saskatchewan is largely affected by both general and local geology at the mine. The general geology consists of three consequential formations: Prairie Evaporite, Dawson Bay, and Winnipegosis formations. The mining conditions are further, and sometimes more prominently, effected by the local geology, consisting of anomalies and specific clay seams.

### 2.1.1 General Geology

The general geology surrounding the mine-ability of potash in Saskatchewan relates to three rock formations. From surface down these are: the Dawson Bay Formation, the Prairie Evaporite Formation, and the Winnipegosis Formation. The potash ore is located in the Prairie Evaporite Formation while the overlying Dawson Bay and underlying Winnipegosis Formations can affect the stability of the mining horizon for various reasons.

#### *2.1.1.1 Dawson Bay Formation*

The Dawson Bay Formation is the layer that directly overlies the potash-bearing Prairie Evaporite Formation and is very important to the stability of potash mining operations. It is used both as a structural member for the potash mining directly below it and, most importantly, as a barrier to incoming brines from upper formations [3]. Because of its importance in keeping out fluids it is imperative that the Dawson Bay formation be kept from fracturing while mining the Prairie Evaporite Formation below.

The Dawson Bay Formation is made up of several different constituent layers and has a geological age in the Middle Devonian (estimated geological age of 380-420 million years) [14]. The primary rock types within the formation are two different limestones: Grey and Brown limestones. At the top of the formation are primarily brown limestones that have much lower strength characteristics. The grey limestones are found throughout the formation [15]. At the bottom of the Dawson Bay, the Second Red Beds member is a composite of anhydrite, dolomite, clays, and calcareous mudstone on top. The general color of the Second Red Beds is a rusty red-brown. [9]. The rock types are important to the structural integrity and water sealing ability of the formation.

The thickness of the Dawson Bay Formation is important to both the structure and the water sealing ability of the formation. Typically, the Dawson Bay is 35 meters thick, with zones up to 60 meters thick. [9] [2]. The thickness combined with the lack of creep present in limestone makes the formation a good structural member for keeping the mined entries stable [4]. The structural integrity of the formation is further impacted by its local composition, as grey limestones can have compressive strengths upwards of 150 MPa, while brown limestone averages approximately 50 MPa [15].

Limestone can be porous so the thickness is important for sealing out sodium-rich brines from the mine workings [15]. When inflows are an issue, grouting can be used to plug up any fractures or natural porosity where inflows are occurring [6]. Even saturated sodium brines are problematic in potash mines because even these will readily dissolve the potassium-rich potash salts. The dissolution weakens the mining back and causes fracturing of the overlying Dawson Bay Formation beam. Areas with an abnormally thin Dawson Bay member could be at risk for instability.

Keeping the Dawson Bay Formation intact is important for the safety and stability of a potash mine. The formation is much stiffer and has a higher yield stress than the underlying, potash bearing, Prairie Evaporite Formation. The extra strength can be used to bear much of the load induced from mining operations. Some mining methods cannot be used in Saskatchewan due to issues created by allowing the Dawson Bay to subside too much. Even with methods that are used, great care must be taken when mining to ensure the strength characteristics of the Dawson Bay are not exceeded.

#### *2.1.1.2 Prairie Evaporite Formation*

The Prairie Evaporite Formation contains the province’s potash reserves. The Prairie Evaporite is a salt formation with several potash-bearing members towards the top of the formation. Each of the four potash-bearing members in Saskatchewan are separated by halite, generally between one and sixty meters thick [6]. The beds are made primarily of two different forms of salt along with one less common salt, so the beds are readily dissolved by water [4]. Also of interest is the depth of the salt formation, something that increases towards the south and reaches depths of 3600 meters in the northern United States [6]. All the potash mining in Saskatchewan is done in the upper part of the Prairie Evaporite Formation where the four potash bearing members are located.

The two upper-most potash bearing members, the Patience Lake and Belle Plaine members, are very similar in composition. In the Saskatoon area, the Patience Lake Member is approximately twenty meters thick [6]. The member does have extensive clay seams that make ground control much more difficult. Overlying the Patience Lake member is a 10-m thick halite beam; underlying the member is a 50-m thick halite seam [16]. Further down, the Belle Plaine member is between two and three meters thick [6]. Halite seams both above and below serve to stabilize the mine openings in these members.

Extensive clay seams give rise to poor ground conditions. The clay seams allow the individual salt beams to slip past each other, reducing the beam's resistance to buckling [6]. The overall material content of the Patience Lake member consists of approximately 5% insoluble material, and 40% to 45% sylvite, the principal potassium mineral salt that is refined into potash, with the remainder of the member consisting of halite [6]. High insoluble content in the Patience Lake and Belle Plaine members reduces the ground stability in the members.

The White Bear member is a minor member of the Prairie Evaporite. The occurrence of the White Bear member in stratigraphic borehole investigations is limited to areas near the Saskatchewan/Manitoba border. No mines actively mine the White Bear member due to the member's limited areal extent [6].

The last potash producing layer is the Esterhazy member, which is of interest in the eastern region of Saskatchewan. This member is typically less than 15 meters thick with a composition more conducive to mining. The member's insoluble content, at approximately one fifth of the amount found in either the Patience Lake or Belle Plaine Members, shows up scattered throughout the member rather than in specific clay seams [6]. The lack of clay seams gives much more favorable ground conditions for mining and allows mines in the east to cut much wider rooms without the need for ground support [9]. Another feature that enhances stability of the Esterhazy member is a 27-m thick salt layer overhead [17]. This salt layer enhances stability by viscously redistributing the stresses induced by mining [8]. The Esterhazy member contains a 5% fraction of carnallite, encountered occasionally as pods, which is a hydrated potassium and magnesium salt that creeps ten times faster than potash and can therefore reduce stability [6]. Because of the few clay seams and thick salt beam above, the Esterhazy has the best ground conditions for mining in the province.

One problem in potash mines is the depths the potash deposits lie at. The depth of the deposits increases towards the south of the province and lie at slightly over one kilometer of depth around Saskatoon, the northernmost area potash is mined [16]. Further to the south, in the region of Regina, the deposits reach depths of 1.5 kilometers. The stresses due to the overburden at depths of even one kilometer are very close to the ultimate compressive strength of potash and well over the 11 MPa yield limit [18]. As a result, excavations in potash cause stress fields that cause the potash to undergo creep strain until all openings eventually close. This creep strain rate is very slow and allows mining safely. Although an issue with potash mining, the openings eventually close and seal so tightly that this has been considered as a way to store radioactive materials long term [19]. The problem for potash mine safety then, is to keep the openings stable for long enough that miners can work safely.

The Prairie Evaporite is the most important formation for mining potash in Saskatchewan because it contains the mining horizon. Most importantly for mining is the quantity, location, and character of clay seams in the mined potash bearing member, as the clay seams greatly reduce the strength of the member.

#### *2.1.1.3 Winnipegosis Formation*

Underneath the Prairie Evaporite formation lies the Winnipegosis formation, made up of primarily carbonates. Lying on the top of the Winnipegosis are many mound structures. It is unknown whether the mounds were created by erosion when they were first laid down, or if the carbonates accumulated around fresh water springs after the Winnipegosis was laid. The mounds are of note because they tend to be associated with anomalous ground conditions in the mining horizon [3].

The mounds, which occur on top of the Winnipegosis formation, are made of several different components and amount to what is effectively a large stalagmite, shown in Figure 2.1 [4]. Most of the mounds are made up of limestone corals and sponges, but the main constituent is limestone [20]. They extend from the top of the Winnipegosis formation up in some cases over

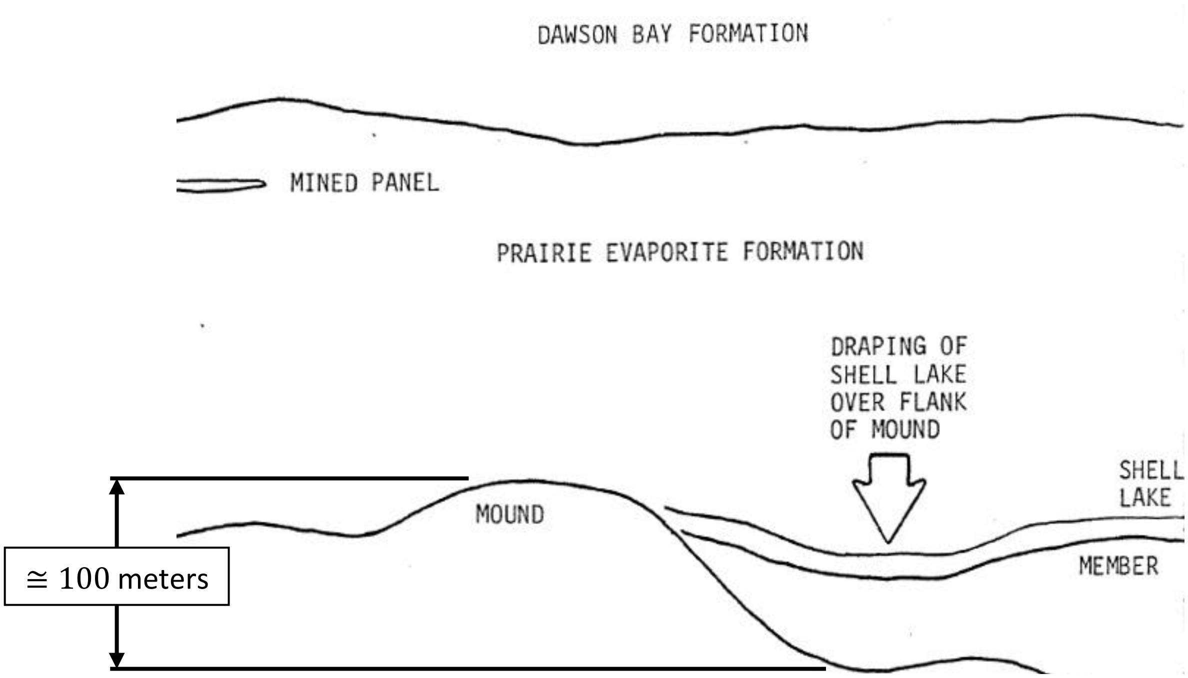



*Figure 2.1 - The Winnipegosis mounds in situ. Printed in the CIM Bulletin, Volume 81, Number 915. Reproduced with permission from the Canadian Institute of Mining, Metallurgy and Petroleum. [2].*

100 meters, or half the height of the Prairie Evaporite formation [6]. Surrounding the cores are large anhydrite halos that are approximately 75% of the height of the core [20].

On top of the mounds is the Shell Lake member, made largely of dolomite and anhydrite [6]. The Shell Lake member sags over the edges of the mounds on the Winnipegosis formation. It indicates that the salt below the Shell Lake member has been dissolved away and in many cases, the sag allows the layers immediately above, including the mining horizon, to sag and even collapse [6].

The Winnipegosis Formation can have an impact on the mining of the Prairie Evaporite Formation. The key features within the formation are the mound structures located on top that are associated with anomalous ground conditions on the mining horizon. The worst ground conditions are located above the flanks of the mounds where the Shell Lake member sags due to salt dissolution below. Anomalies create lower stability conditions that require remedial actions, such as the current anomalous ground bolting procedure, to provide adequate stability.

### 2.1.2 Local Geological Effects on Potash Mining Methods

Even more important to potash mining is the local geology and material strengths. Potash is a soft rock that displays creep strain at stresses above approximately 25% of the unconfined compressive strength, and mine layouts must be designed accordingly [21]. The strength of the Dawson Bay Formation is also important for several reasons. Local geological features, such as anomalies, can greatly affect mining operations and the safety of the mine. The three main types of anomalies are washouts, leach anomalies, and collapse structures [16]. Salt anomalies indicate a much higher potential for brine inflows that would threaten the stability of the mine [22]. The worst of the three for mine ground conditions are collapse anomalies, one of which is shown in Figure 2.2, and should be avoided wherever possible.

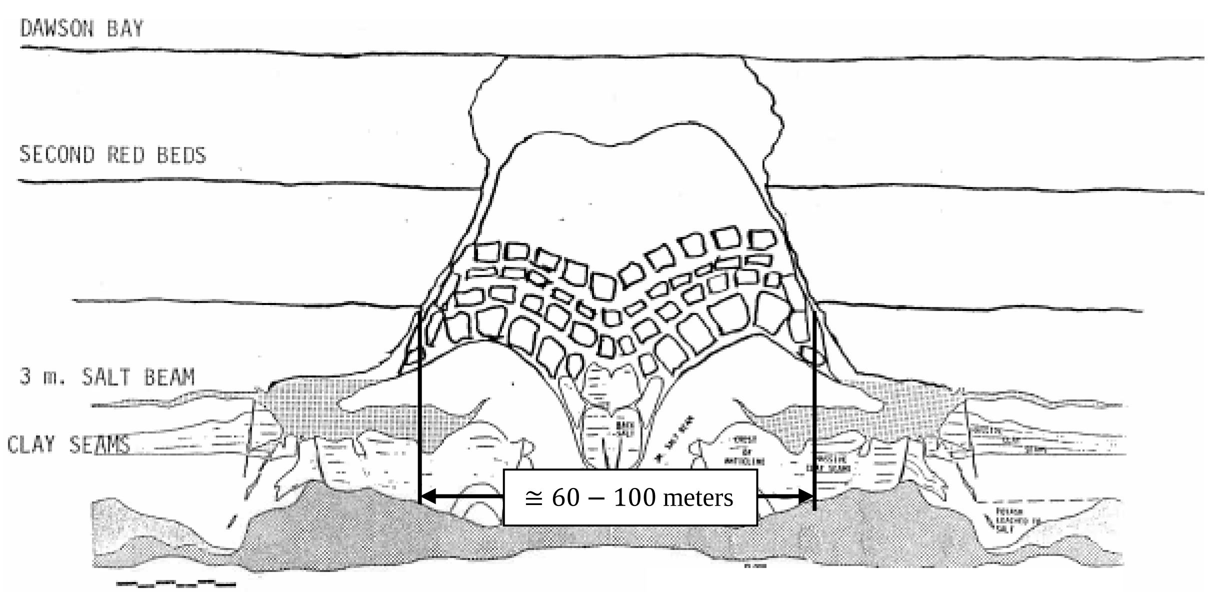



*Figure 2.2 - The cross-sectional view of the structure of a typical collapse anomaly. Printed in the CIM Bulletin, Volume 81, Number 915. Reproduced with permission from the Canadian Institute of Mining, Metallurgy and Petroleum. [2].*

#### *2.1.2.1 Strength of Potash*

The strength of potash is integral to the stability of the entire mine. Once rooms of potash are removed, pillars of potash remain to ensure back stability: the strength of the potash is of primary importance when determining the design size of the pillars. The strength, shape, and material all govern how quickly the openings will fail via creep. The potash salt tends to creep one order of magnitude slower than the carnallite, sometimes found in the potash beds, and one order of magnitude faster than that of the halite salts [6]. Higher creep rates result in higher closure rates and long-term instability. Mine stability is affected by both the strength and the creep rates of the material in the pillars.

Potash salts behave differently in lab tests, where they are largely unconfined, then they do when underground in a mining operation, where they are confined. Several formulas for the creep strain rates have been developed, both from triaxial tests and from in field measurements [23] [25]. In potash mines, the actual strain rate slows over time [8]. It is believed the effect of the pillars widening over time due to creep is the reason behind this observation [24].

In all Saskatchewan mine cases, potash pillars will be stressed beyond their unconfined strength limits. In any mine, the unconfined compressive strength (UCS) of the potash will be at least temporarily exceeded if the mine depth exceeds approximately 500 meters given an average UCS of potash of 28 MPa [7] [8]. Because of the total weight of the overburden, all pillars will eventually yield in Saskatchewan to relieve the excess stresses present in as mined pillars.

When rooms are mined in any rock type the stresses originally present in the mined-out rock must be redistributed into the surrounding pillars. In potash, if the horizontal extent of a panel exceeds the depth of the mine, it is not possible for all the stresses to be redistributed to the undisturbed flanking pillars [8]. For this reason, panels in Saskatchewan are designed small enough that the majority of the stresses can be redistributed to the flanking pillars. While extraction ratios may approach 50% with in a mining panel, overall extraction ratios are kept low enough to allow stress redistribution to much larger abutment pillars. If the back has a thick salt beam in it the stresses will be redistributed uniformly, greatly reducing any excess stress gradients and increasing stability of the mined openings [8]. Uniform redistribution causes all mined rooms in a panel to close at approximately the same rate [8]. In Saskatchewan, typical closure rates are in the range of

2 to 3 centimeters per 100 days [26]. Utilizing viscous stress redistribution from thick salt beams and restricted extraction ratios ensures the safety of personnel in the openings.

#### *2.1.2.2 Strength of the Dawson Bay Formation*

The Dawson Bay limestone acts as a structural member so it is important to note the strength characteristics of this formation. The limestones making up the Dawson Bay Formation are both much stronger and stiffer than the potash underneath making the Dawson Bay an important structural member [27]. Safe mining practices are designed to prevent the Dawson Bay Formation from exceeding its strength properties.

The geometry of the mined rooms changes the size and locations of the maximum stresses in the member. The maximum compressive stress is located over the edge of the mined rooms. According to numerical simulations of 13 to 14 mined rooms, the limestone settles approximately 0.52 meters in the first 1200 days and 0.61 meters over the first 2000 days at the center of the mining panel [9]. Mine planning therefore, can greatly reduce the chance of fracturing the Dawson Bay Formation.

#### *2.1.2.3 The Effect of Anomalies on Salt Mining*

One of the major concerns for Saskatchewan potash mines are anomalies on or near the mining horizon. The main cause of such anomalies is thought to be sodium rich brines infiltrating potash laden beds, dissolving the potash and replacing it with halite [2]. Water inflows, which happen much more frequently around anomalies than elsewhere, can cause bad ground conditions and high closure rates. In some cases, mines can be closed due to flooding from these inflows and the anomalies should be avoided altogether [28].

Leach anomalies are the most common type of anomaly as they may occur on their own but also typically surround collapse anomalies [2]. Leach anomalies occur when the potash salts are dissolved out of the main potash layers and are replaced by halite without disturbing the bedding sequence of the Prairie Evaporite Formation [16]. The presence of water at the time the anomaly was created results in a higher chance of water inflows in these areas than in other areas of the mine.

Collapse anomalies can be large structures, with around 60 to 120 meter-wide collapse structures at the center [2]. At the then Canadian Central Potash (CCP) Colonsay mine, now operated by Mosaic, the frequency of collapse structures was approximately one collapse for every nine square kilometers mined [2]. Collapse structures are typically surrounded by leach anomalies. The resulting risk of water from collapse anomalies is higher than that of a leach anomaly, making the collapse anomaly more hazardous to mining operations.

When mining, collapse structures can be indicated by a rapid drop in elevation of the potash beds in the mining horizon. The drop is typically indicative of the leach anomaly that will surround most collapse structures. Potash beds may be able to drop up to 30 meters when a leach anomaly is present without there being a collapse structure present [2]. Typically, multiple clay seams present can merge into one solid thick beam of clay as the salt normally between the layers is dissolved away around these structures; the merged clay seam can sometimes be up to 1.5 meters thick [2]. The actual collapse structure in the center is a void created by the dissolution of the salt beds and may be filled with Dawson Bay Formation limestones that have collapsed from above [6]. Other 3-D seismic reflection techniques are now typically used to find these structures without mining near them. Monitoring for the presence of collapse structures, and typically avoiding them, is integral to maintain stability in the mine openings.

The last type of salt anomaly is the washout anomaly. These were likely caused by rain or other water currents at the time of the salt beds deposition [16]. These anomalies are not usually a concern for potash mines.

Most anomalies encountered on the mining horizon are dangerous to the stability of the mine openings. The most hazardous is the collapse anomaly that may be filled by limestone from the overlying Dawson Bay Formation. The voids left in the ground from collapse anomalies are usually laden with water that may invade the potash mine workings if cut into. To detect the anomalies before they cause problems in the mine, mines use a variety of indicators including the presence of leach anomalies. Most areas are mapped using various techniques before being mined to make sure no anomalies hazardous to the mine are present. Avoiding the anomalies, and specifically collapse anomalies, reduces the chance of water inflows that could flood the mine. The lesser hazards posed by leach and washout anomalies are typically not mine threatening, and can still typically be mined if the stability is improved through the use of rock bolts or other means, discussed later.

## 2.2 Potash Mining Methods in Saskatchewan

For a mining method to be viable in Saskatchewan potash mines, the method must be able to mitigate the challenges inherent to the ground conditions in Saskatchewan. The three main methods used are long room and pillar, stress relief mining, and solution mining. As discussed below, each mining method is best suited for certain ground conditions that may be found in the province.

When the ore body is excavated, the *in-situ* stresses must be redistributed. This redistribution of stresses is key in determining the effectiveness of different mining methods. Excavation openings will cause stresses to redistribute between 4 and 10 openings wide [8]. Due to the depths inherent to Saskatchewan mines, it is necessary to leave abutment pillars at the edges of mined panels that are wide enough to support the redistributed load, and therefore prevent the in panel pillars from completely failing, something that could potentially allow the overlying Dawson Bay Formation to fracture and cause a water inflow [8]. The exact distribution of the stresses is dependent on the mining method used.

The determination of what mining method to use in a potash mine is affected mainly by geology, and depth. Each method then takes the inherent stresses and redistributes them differently via mining patterns and opening geometry. Dry mining techniques are typically used where ground conditions allow. Where the depth is too great to mine via dry techniques safely, solution mining can be employed. Employing the correct mining method helps to ensure the safety of the mine personnel and openings.

### 2.2.1 Long Room and Pillar Dry Mining

Long room and pillar mining is a technique used in eastern Saskatchewan potash mines as well as some mines in the Saskatoon area, such as PCS Lanigan. It is most conducive to areas that have low clay content and more importantly, have few clay seams that create thin, inherently less stable, salt beams in the mine back. It is, however, a simple cutting pattern with very few corners. Due to lower clay content, and a lack of distinct clay seams, the eastern Saskatchewan mines can cut rooms that are much wider than those in the Saskatoon area [9]. The boring machines used in the east are almost exclusively 2.5 meters tall and 8 meters wide utilizing four rotors [9]. Long room and pillar mining allows simple and efficient extraction of the potash in good ground conditions.

Long room and pillar mining is performed in a simple sequence. Before mining, a block of ore is outlined by 4 or 5 development entries. The block is then mined by cutting entries straight from one end of the block to the other [4]. Several passes are done on each mining room before a pillar is left to ensure stability of the mine back. In eastern mines, the rooms are typically cut approximately 18 meters wide before a pillar is left [9]. In this sequence, pillars are left indefinitely and represent ore that is never recovered. The extraction ratios for this mining method are approximately 35% to 40% with some individual blocks, otherwise known as panels, reaching 50% [29]. Mining progresses by sequentially cutting panels and repeating this process.

Long room and pillar mining of the potash ore represents the simplest possible way to deal with the long-term induced mining stresses. In short, the pillars are designed such that they carry the weight of the overlying rock [8]. The way the stresses concentrate in the pillars is shown in Figure 2.3. With the potash already loaded to near its unconfined compressive strength in the mining horizon, it is unreasonable to simply divide the overburden weight for an area by the cross-sectional area of the pillars left such that they are not loaded beyond the UCS of potash [8]. It is

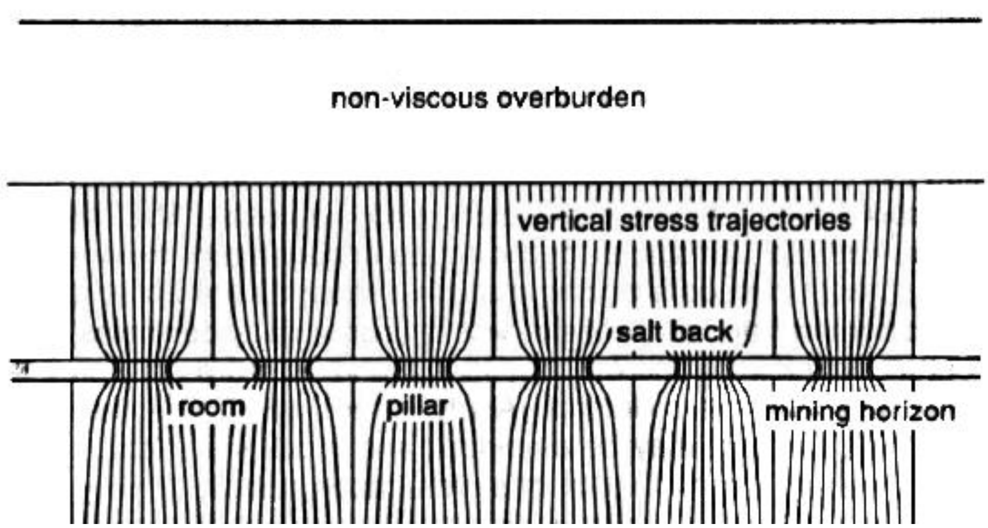



*Figure 2.3 - The stress trajectories in a cross section of a long room and pillar mine. The stress is seen to concentrate in the pillars between the rooms on the mining horizon. Reproduced by permission of Dusseault, [8].*

therefore understood that the pillars be designed such that they have to yield and relieve the stress induced by mining, a technique known as designing the pillars in a failed condition. The UCS is exceeded in any Saskatchewan mine due to the strength of potash being only marginally higher than the induced stress at the depths it is mined. The potash must be allowed to creep until a new equilibrium is found; this equilibrium is found when mine openings have completely closed in Saskatchewan mines.

The long-term stability of the mines is achieved both by pillar widening and pillar confinement, and must take into account the yield and creep of potash ore [8]. Figure 2.4 demonstrates pillar confinement. Confinement restricts the ability of the potash to buckle, which in turn greatly increases the maximum stresses that the pillar can withstand. Long pillars have greater confinement because they restrict the movement of the potash along the length of the pillar [8]. The widening of pillars by creep flow further increase the load carrying capacity of the pillars over that achieved via confinement.

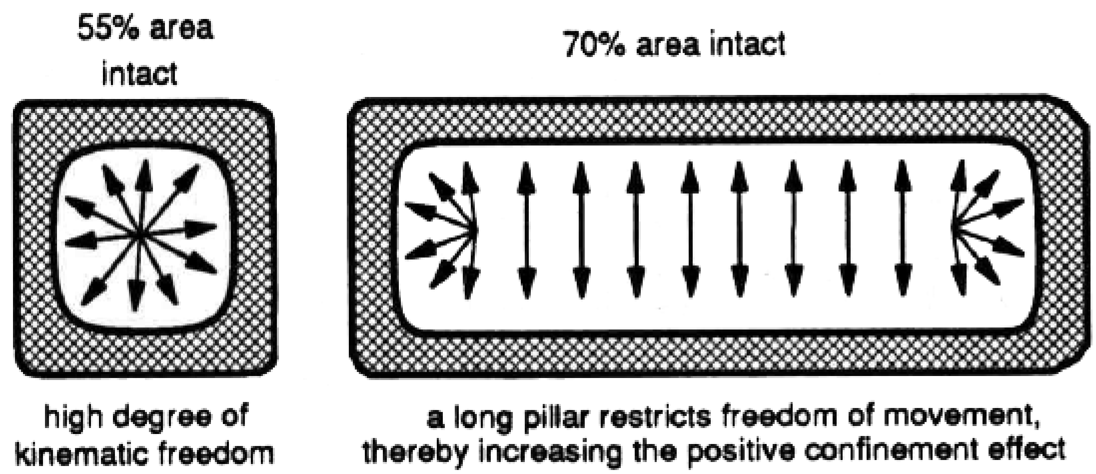



*Figure 2.4 - The effect of long pillars on the confinement of the potash. Confinement in pillars increases the load at which the pillar will fail. Hatched area represents area that is excessively yielded. Reproduced by permission of Dusseault, [8].*

Long room and pillar mining is a technique commonly used in Saskatchewan potash mines. In regions with only moderate depths and good ground conditions, the system is preferred because it has very few corners and is simple to both develop and plan. In regions around Saskatoon, where the ground conditions are less favorable, due to extensive clay seams, this method does not work as well because of reduced stability compared to other methods. Long room and pillar mining is the method of choice in eastern Saskatchewan potash mines.

### 2.2.2 Stress Relief Mining

In the area around Saskatoon a more common method of potash mining is stress relief mining. The Patience Lake member, which is the member of interest for mines in the Saskatoon area, contains clay seams not present in other members. The clay seams result in less stable mine openings [6]. Stress relief mining is a method designed to allow mining in worse ground conditions and greater depths [28]. The technique uses four or five entries running beside each other to protect the center development headings and sacrifice the outside stress ways, or production headings [9]. A five-entry pattern cross section is shown in Figure 2.5. The typical mine pattern, known as a chevron or herringbone pattern, is also pictured in Figure 2.5. Fewer development headings are required here than with long room and pillar mining. Only two or three centrally located development headings are required rather than outlining the whole block with development headings as in long room and pillar mining [9].

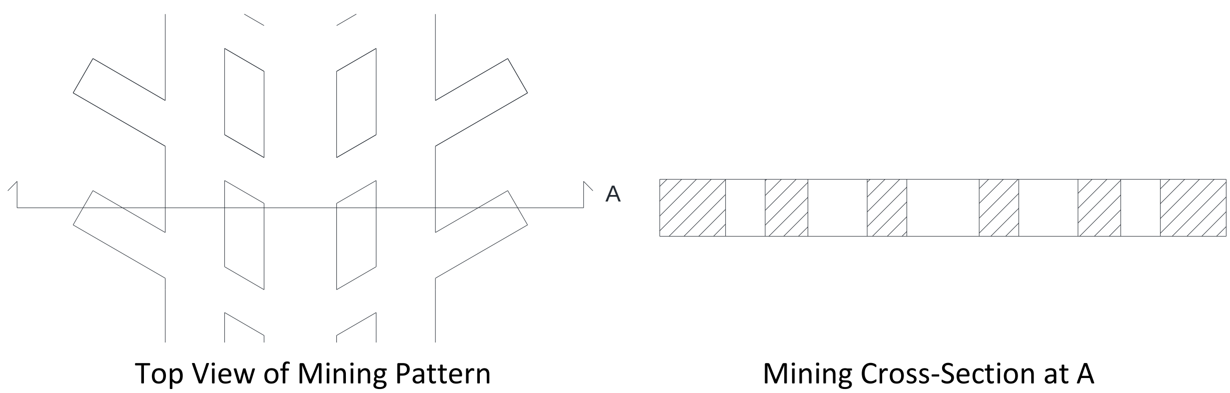



*Figure 2.5 - The chevron, or herringbone, pattern typically used in stress relief mining. A cross section reveals a 5-entry stress relief pattern with a single main development heading in the center.*

As depicted in Figure 2.6, the vertical stresses within the pillars of a stress relief mining pattern are greatly reduced as they shed stresses to the barrier pillars. The goal of stress relief mining is to allow the outer rooms (production rooms) to sag and collapse. The failed area, due to excess sagging, results in the stresses being shed to adjacent abutments [9]. The greatest stress reduction is in development headings, located in the middle of the pattern. This stress reduction allows the travel-ways and transport headings to remain relatively stable when mining in high stress ground. Great care must be taken when designing the cutting pattern to be used in such a mine as stress ways that are too wide will result in rapid back deterioration in all adjacent mine headings [17]. Stress reduction allows for the mining of deeper potash deposits and deposits with extensive clay seams that are inherently less stable.

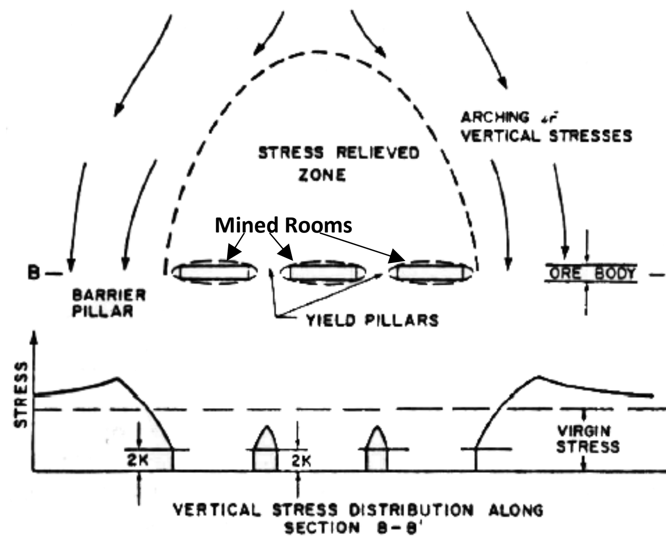



*Figure 2.6 - The stress relieved zone in a stress relief mining method. This shows how the vertical stresses are relieved when the yield pillars give way. This effect is caused by the center portion of the mining pattern being able to deform more freely than the outer edges of the pattern. The vertical stresses typically do not cause failure because failure tends to occur in the horizontally oriented salt beams, where the rock mass is weakest, due to horizontal stresses. Reproduced by permission, [8].*

The more destructive horizontal stresses are relieved in a similar fashion [30]. The near hydro-static nature of the stress state in potash material results in approximately the same stresses horizontally as are present vertically. While the vertical stresses are supported by abutment pillars and inter-panel pillars that are squat and therefore are not prone to buckling, the horizontal stresses act on the salt beams in the mine back. The salt beams in the mine back are much thinner than the vertical pillars and more prone to failure. Buckling of the mine back because of the horizontal stresses is the typical failure mode of rooms not using stress relief methods. As the horizontal stresses concentrate around the top and bottom corners of the outside rooms, as depicted in Figure 2.7, the salt beams in this creep or fracture [30]. The creep and failure of the back in this area reduces the stiffness of the area immediately around the opening and sheds the stress to more

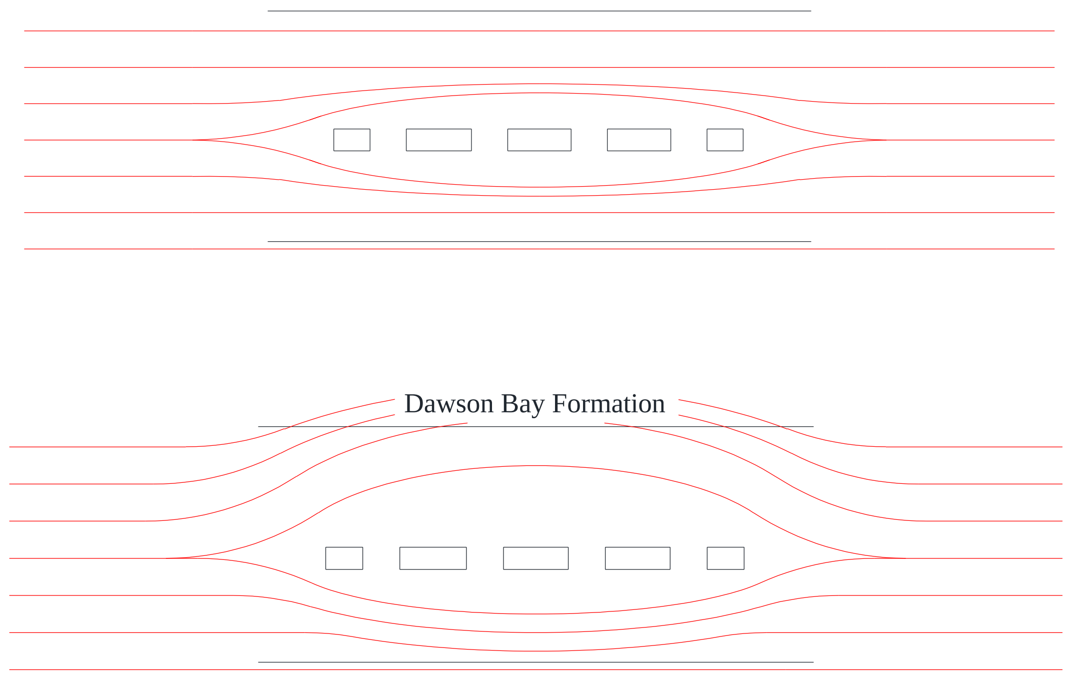



*Figure 2.7 - Horizontal stress patterns around stress relief entries. Stress trajectories are shown in red. Shown top is how stress redistribute in a homogenous ore-body. As back and floor failure occur around the outside rooms, the stresses are relieved and spread further from the mine back near center excavations. Stresses may be further reduced in the mine back in Saskatchewan by utilizing the Dawson Bay Formation as a structural member. The limestones making up the Dawson Bay Formation are much stiffer than the potash ore and therefore take more of the load than the potash beds, relieving the stress and preserving the center entries.*

competent beds further from the opening, such as the Dawson Bay Formation. The back and floor around excavations in the middle of the mining pattern experience much lower horizontal stresses and are therefore protected. This results in the middle excavations being more stable and this makes them last longer.

Stress relief mining extraction ratios are like those of long room and pillar mining. Mines in the Saskatoon area are limited to cutting rooms that are between 10 and 12 meters wide as the mine back is unsatisfactorily unstable otherwise. The resultant extraction ratios are similar to the long room and pillar mining extraction ratios of between 35% and 40% with some individual panels reaching 50% [28]. The extraction ratios of both dry mining techniques are inherently low.

In the Patience Lake Member, where there is a higher clay content than in other members, stress relief mining is often used. The technique is designed to reduce the stresses placed on the rock surrounding the development headings in the center of a panel. Due to the higher insoluble content in the Patience Lake Member, reducing the stress around these openings is necessary. In other areas with moderate depth deposits that are too deep for long room and pillar mining, the stress relief method can be used. The stress relief mining method is an improvement over long room and pillar mining in terms of the deposits that are mineable, even though the system is a more complicated layout.

## 2.3 Ground Support Materials and Systems

In normal ground conditions, the ground is stabilized sufficiently via mining methods and cutting patterns that no additional ground support is necessary. In some ground conditions however, mining methods alone don't provide the level of stability required for underground personnel in the mine workings. Such ground conditions include any anomalies that may result in thinned salt beams or clay seams intersecting the mine back, as well as low hanging clay seams created by undulation or by the boring machine cutting too high. Additional ground support and remedial actions can be used to improve the ground stability in these situations. Most common in potash mines are tendon type ground supports, mainly comprised of various bolt styles. Installing these bolts, which may be longer than the excavation is tall, requires specialized machinery.

### 2.3.1 Tendon Type Ground Supports

There are many unique styles of rock bolts used in underground mines. Each of the different bolt styles vary from shaft style to anchoring method. The three most common bolts are mechanically anchored rock bolts, resin rebar bolts, and grouted cable bolts. Other styles of bolts include Swellex® style rock bolts, friction stabilizers, stelpipe style grout-able rock bolts, and modified cone rock bolts. Due to the specific requirements of potash mining, including that the bolts must be able to stretch considerably before breaking, point-anchored bolts are the most common type used near the mining face when anomalous ground conditions exist. Point anchored bolts can strain along their entire length unlike bolts that are anchored along their full length, such as resin rebar or cable bolts. There is a further need for the bolts to immediately support load for productivity reasons. For these reasons, the mechanically anchored bolt is generally the only style of bolt used for the high movement area near the mining face in Saskatchewan potash mines.

In sedimentary mines, such as the Saskatchewan potash mines, the bolts are used to serve two purposes, to prevent fractured potash blocks from sliding out of the mine back and into the mine workings, and to increase the overall stability of the mined entries. Rock bolts are installed across multiple sedimentary layers and work to pin the bedded layers together across planes of weakness such as those shown previously in Figure 1.1. The bolts can carry the deadweight of potash blocks that may fracture and therefore prevent them from falling into the mined entry. Pinning the sedimentary layers together also serves to create a thicker, more structurally competent beam that can better resist buckling due to horizontal stresses. The bolts are installed in areas where otherwise reduced ground stability would prevent the safe excavation of an entry.

#### *2.3.1.1 Mechanically Anchored Rock Bolts*

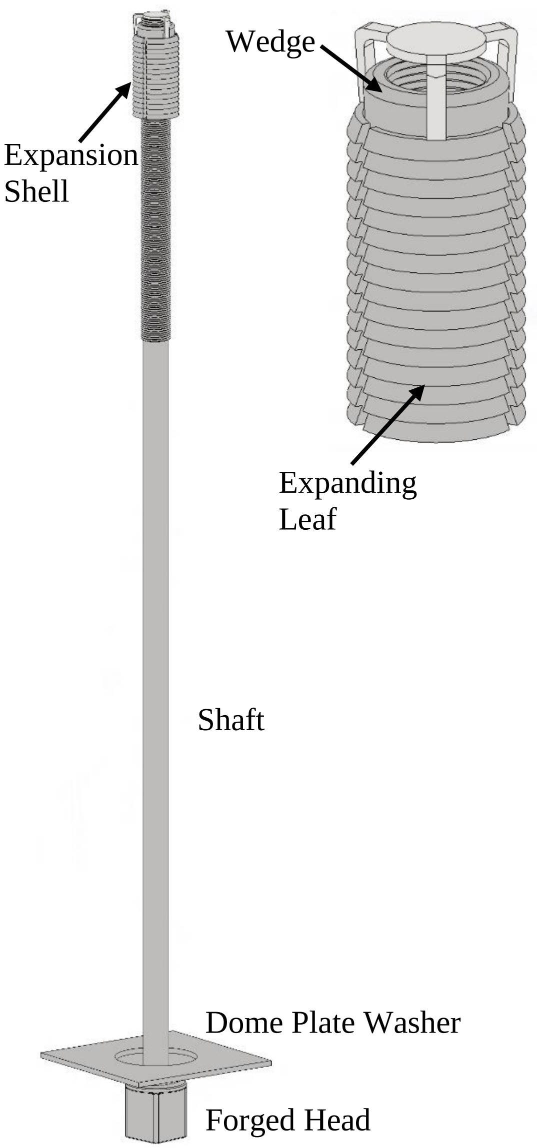



*Figure 2.8 - A three-leaf mechanical expansion shell rock bolt. When the head is turned, the wedge is drawn through the leaves. This causes the leaves to be forced outward into the rock to lock the bolt in place.*

Mechanical rock bolts are a widely used bolt across many types of mines. The name is derived from the point anchor used to fasten the bolts. Mechanical bolts are very simple to install due to their design, and the process is easily automated making the bolt useful in a wide range of mines. Because the bolt is anchored via a point anchor at the end of the bolt, it can elongate more and accommodate shifting ground conditions better than a bolt that is anchored along its full length; with typical elongation at break of between 15 and 20 percent for steel, a two-meter-long mechanical bolt may be able to deform 40 cm or more before total failure. A similar length fully anchored bolt will only deform in the region of a fracture, resulting in deformation that may only reach millimeters. The design of the bolt makes it suitable for Saskatchewan potash mines.

The mechanically anchored rock bolt is a simple roof support device that is equally simple to install. The bolt, as pictured in Figure 2.8, is made of four main parts: the shaft, the expansion shell, the head, and the bearing plate or washer. The shafts for these bolts are currently made of a steel or fiberglass rod, the fiberglass used to eliminate corrosion. When inserted into the hole, the head is turned with respect to the expansion shell causing the shell to expand and lock to the surrounding rock. These bolts have some approximate load indication depending on the bearing plate used. If the plate used is domed, the way the plate deforms will give a visual indication of the load on the bolt [11]. The torque applied to the bolt can also be correlated to the tension on the bolt. The design creates a point anchor system for the bolt that is different than most other bolt styles.

It is important for these bolts to be installed perpendicular to the mine opening. The main advantage of the mechanical bolt is that it is an active support type: it is always tensioned. The downside to active support is that the bolts are much less effective if inserted sub-perpendicular to the face. Due to twisting and other factors, the bolts will lose up to 75% of the original tension if they are inserted at 30 degrees from perpendicular to the opening [11]. Further losses in load capacity can occur if the washer plate does not have even contact with ground or if the expansion shell is installed in weak ground. Perpendicular bolts provide the highest support forces on rock slabs and are therefore most effective.

Many potash mines in Saskatchewan utilize mechanical rock bolts extensively. Pre-tensioning is a must to do this effectively. The cost of the bolts, as well as the simplicity of automating the installation, has also lead to their widespread use. Many of the shortcomings associated with the bolts are not an issue in Saskatchewan. For example, corrosion resistance is not important because although the bolt is surrounded by salt, there is no water in the potash mine workings and the bolts are therefore not prone to rusting. These bolts are ideal in Saskatchewan potash mines for immediate roof support to ensure safety of mining personnel near mine faces where large deformations are expected.

### 2.3.2 Rock Bolting Machinery

Installing rock bolts in underground mine environments requires specialized machinery. The currently available machines used for this purpose come in two main configurations: dedicated bolting machines, and bolter-miner set ups. Although not specifically used in mining applications, tunnel boring machines also install some different ground support types. Choosing the best method of installation for a specific mine can contribute to increased mine efficiency and safety. To make the best choice for a mine, it is important to understand the advantages of each type of design.

### 2.3.2.1 Dedicated Bolting Machines

Dedicated bolting machines are made by most underground equipment manufacturers and, while each one has distinct characteristics, they all have the same general components. They are typically made up of an operator's cab, and a telescoping boom with a drilling mast on the end. The drilling mast is used to house the bolts and drilling systems. The machines are generally diesel or electrically driven. The DynaBolter®, pictured in Figure 2.9, follows a typical, single bolting mast, machine design, much like those used by PotashCorp.

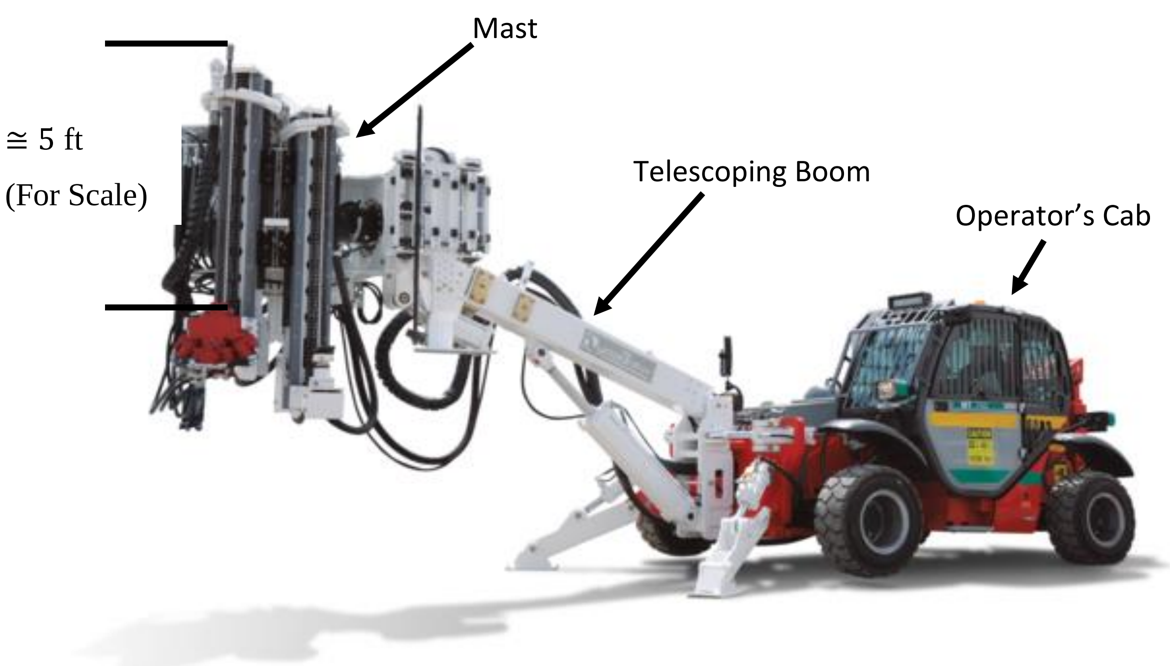



*Figure 2.9 - The DynaBolter® V5.0. The design is typical of most dedicated bolting machines. Reproduced by permission of DynaIndustrial LP, [38].*

Bolting machines are designed for the exclusive purpose of installing roof bolts and serve no other function. They are built such that they have greater versatility in the types of bolts and support systems that can be installed. These machines must either work behind the boring machines or have the boring machines moved to have access to the mine back near the active face. If the boring machines are left in place and the bolter works behind, there is a long stretch of ground which is completely unsupported and this may not be acceptable from a safety standpoint. If the boring machines are backed out of the tunnel to allow the bolting machines greater access, there is considerable time wasted. Boring machines are slow so any maneuvering required is time-expensive [31]. Generally, these machines require mining to stop in the area for bolts to be installed.

The specialized bolting machines that are available offer several features that are advantageous from a productivity standpoint. Some of the machines available will offer multiple bolting heads on the same machine in order allow the installation of multiple bolts at once [32] [33]. This may allow bolting to go two or more times faster than a single bolting head and can save considerable time. Some dedicated bolting machines offer additional features over top of simply bolting such as meshing rollers [34]. Bolt carousels that allow the machine to carry more than one bolt at a time may also help to increase the bolting speed [32]. Another advantage of the specialized bolting machines is that many can install many unique styles of bolts simply by changing the equipment installed on the drilling mast [35]. The versatility of the specialized bolting machine has led to its widespread use in both hard and soft rock mines.

Bolting machines can also be made in a different configuration that is much more labor intensive for installing the bolts. This second style of bolter involves a cab with a deck on a scissor lift such as the MacLean scissor bolters [35]. Mining personnel stand on the deck with hand held rock drills and install the bolts manually. This style of bolting machine is an old and labor intensive system as it involves having mining personnel in unsupported ground while installing bolts. Sometimes such a machine will have a shield overhead with slots cut in it to allow bolting through the shield. This method is typically not used in potash mines.

#### *2.3.2.2 Bolter-Borer Configurations*

To offer an improved productivity system to underground mines, manufacturers have added bolting heads to boring machines so that one machine can both mine and install bolts. This configuration eliminates the need for the machines to be maneuvered as would be the case with a specialized bolting machine.

Currently the only style of machine that has been outfitted with such a bolting system is the non-continuous variety of miners such as the one shown in Figure 2.10. These miners are built with a rotating drum that is forced into the end of the tunnel by means of a telescoping boom. The drum digs into the tunnel with the chassis stationary until the telescoping boom can no longer reach, at which point the boom is retracted, the chassis advanced down the tunnel and the process is repeated. This style of boring machine is conducive to adding bolt functionality since the chassis is stationary for most of the mining cycle. On a continuous miner, the constant relative movement of the chassis and bolt installation locations makes installation more difficult. Non-continuous boring machines still must stop mining to advance the chassis but can improve productivity in poor ground because multiple machines are not required.

Many of these machines have multiple bolting heads attached to them. The idea behind this is that the bolting heads never have to move to bolt across the breadth of the tunnel. This serves to speed up the bolting process and thereby does not hinder the boring machine. The miner-bolter setup may be designed to be able to bolt and mine at the same time, further speeding up the process. Adding multiple bolting heads allows each bolting head to install bolts in a line, rather than installing multiple bolts across the width of the tunnel, to speed up mining.

*Figure 2.10 - The JoyGlobal 12ED30 miner-bolter. The system is designed to force the drum on the right into the end of the tunnel for digging. Installing multiple bolt heads on this machine is fairly simple as the chassis is stationary most of the time. Reproduced by permission of Komatsu Mining Group, [31].*

Ideal process times are not achieved via either the specialized bolter or current miner-bolter systems for Saskatchewan potash mining applications. Current processes utilizing a combination of mining and bolting machines suffer from poor advancement rates in poor ground conditions, where bolting is required, but realize very efficient ore extraction in areas where bolting is not required. On the contrary, current borer-bolter systems are not capable of the same advance rates in good ground conditions, where bolts are not required, as current continuous boring systems but the process rates are not slowed by anomalous ground conditions. The machines are slower in good ground conditions because they are built on non-continuous boring machine chassis. Ideal process times would be achieved by using a continuous boring machine in good ground conditions, outfitted with bolting equipment for poor ground conditions. It is currently unknown quantitatively how each of the outlined machines perform for average advancement rates and room stability when compared to other outlined processes.

# Chapter 3 Research Goals

Current mining methods in potash mines are slow in anomalous ground conditions because of the need to increase stability by bolting. It is thought that there is room to improve both productivity and safety by modifying the machinery used to mine in these conditions. The goal of this research was to quantify the potential for increased safety and productivity when mining in anomalous ground conditions with a continuous miner-bolter system in Saskatchewan potash mines. Advances in the production rate and safety levels would be advantageous to the potash industry in Saskatchewan.

The goal of this research, therefore, was to determine the feasibility of using a bolting machine mounted on the currently used boring machines to achieve improved safety and performance of the mining procedure in anomalous ground conditions. The study has been broken into three sub-studies:

1. A study on the time required to install bolts in the mine back as well as the amount of time that can be saved in the current process. This included finding the amount of time expected to be saved via several different proposed procedures and the difference in productivity between each. This required a full uncertainty analysis.

2. Determination of the expected stability for several proposed replacement bolting methods. The safety of the proposed procedure must be compared to current safe practices to determine the acceptability of the proposed methods.

3. The design of a prototype machine concept to demonstrate the feasibility of implementing the proposed procedure. The geometry of the prototype machine was used to determine reasonable procedures and geometries for the other two studies.

Upon completion of these three sub-studies it was possible to determine the feasibility for such a bolting system, the safety of mining in the revised mining and bolting procedure, and the possibility of such a system from a design standpoint.

# Chapter 4 Productivity Modelling of Current and Proposed Practice

When mining personnel encounter anomalous ground conditions underground, it is assumed that the stability of the ground is reduced. Bolts are installed in the back to maintain the stability of the ground for the safety of personnel. These bolts improve stability by pinning layers of rock together to create larger effective beams in the roof as well as reducing the chance of a salt block fracturing and sliding out of the back. The bolts further improve safety in the mine workings by potentially holding the weight of any potash blocks that do separate, preventing them from falling into the workings. There are a few important pieces of equipment and other components that affect the installation of the rock bolts. Installing them also requires a different mining procedure, referred to as the anomalous ground bolting procedure, from the normal ground condition mining procedure, which does not require any bolts to be installed.

There are two separate pieces of equipment presently used to complete the anomalous ground bolting procedure. The first piece is the boring machine used to drive the tunnel. The borers used at the host mine are two-rotor continuous style borers. This style of borer is always in motion while boring the tunnel, unlike non-continuous borers that only move the cutting head. Given the limited head space above the boring machines, installing bolts up to the mining face requires that the boring machine be moved from the end of the tunnel to allow access for a specialized bolting machine. The bolting machines are expressly designed to install bolts and serve no other purpose but do install a wide variety of rock bolts. While both machines can do individual jobs very well, using both in the same tunnel results in downtime when moving and changing the various mining machines.

## 4.1 Time-Consuming Tasks

There are generally four components to the anomalous ground bolting procedure that require considerable amounts of time. These tasks are: tunnel boring, movement of bolting and boring machines, brattice installation, and bolt installation. Both brattice installation, used for ventilation purposes, and machine movement time amount to a smaller portion of the overall time spent mining. Due to the number of bolts installed in the current bolting pattern, shown in Figure 4.1, bolt installation time, at approximately one minute per bolt, can amount to a considerable amount of the mining process. Mining in normal ground is considerably faster than mining anomalous ground because bolts are not required and therefore the bolt installation time is eliminated, while other parts of the procedure are greatly reduced.

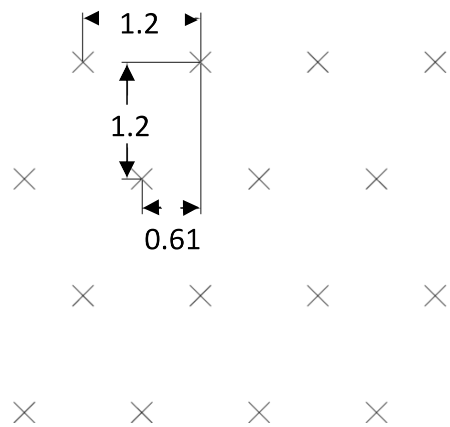



*Figure 4.1 - The currently used 1.2 meter spacing staggered bolting pattern. Dimensions shown are expressed in meters. Alternative proposed patterns include changing from a staggered pattern to a simple square and reducing the spacing to as little as 0.91 meters.*

Ventilation brattice is more of a concern in single pass rooms than in two-pass rooms. Brattice is effectively a plastic curtain installed down the middle of the tunnel with Hilti® pins to facilitate the ventilation of the end of a mined tunnel. Fresh air is pumped in down one side and past the operator before being exhausted down the other side of the curtain. A diagram is shown in Figure 4.2. In a room that is two passes wide, it is possible to fit the boring machine and bolting machine side by side in the tunnel with the brattice curtain in between. This means that much less of the brattice must be pulled down and reinstalled every time machines are changed than in a single pass room where this is not the case. As can be seen in Figure 4.2, in a single pass room the borer takes up the full width of the tunnel. For this reason, the boring machine must be taken all the way out of the entry to allow the bolting machine in and the brattice must be taken down to allow this to happen. During the anomalous ground bolting procedure, the machines must be changed every eight meters of tunnel advance, requiring the brattice to often be pulled down and reinstalled as well.

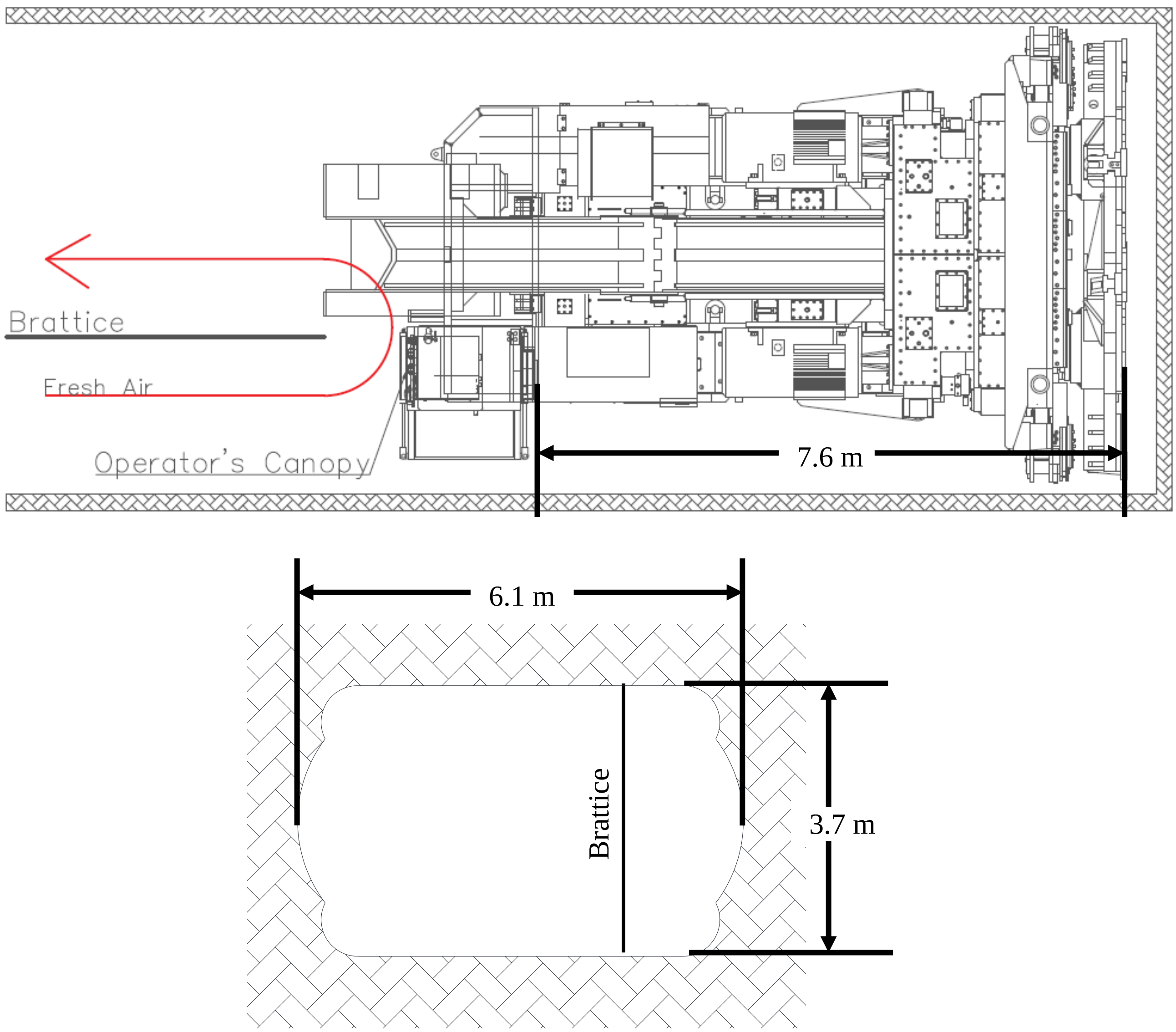



*Figure 4.2 - Brattice in the context of a single pass mined tunnel. Both top view (top), and cross-section view (bottom) are shown. The purpose of brattice is to provide fresh air up to the end of a tunnel where air would not naturally go otherwise. The conveyor system would run approximately down the middle of the tunnel. The fresh air is generally sent down the right side of the brattice as seen in the figure because dust from the conveyor system would make it difficult to breathe should the air go the other direction. Reproduced by permission of PotashCorp.*

Machine movement time is also longer in single pass rooms than in two-pass wide rooms. The boring machines can move at only several kilometers per hour which makes moving a long distances down the tunnel a time-consuming endeavor. In a two-pass room, because the borer can pass beside the bolting machine in the entry, the borer only needs to be backed up approximately 16 meters to allow the bolting machine roof access. In single pass rooms, the borer must be completely removed from the tunnel after cutting every approximately eight meters, for the full tunnel length of up to approximately sixty-one meters in length as this is the maximum extension of the flexible conveyor system. Near the end of the tunnel this can result in several minutes used travelling in each direction with the boring machine. In some cases, the mines would consider using a procedure for a two-pass room in anomalous ground to avoid the additional machine movement time.

Bolt installation time is the greatest use of time apart from tunnel boring. For a two-pass room, the bolt pattern used results in seven bolts installed for every 1.2 meters of borer advancement. Along the length of a tunnel that may be 61-m in length or more, this results in a total of approximately 370 bolts installed in the mine back. The rows of bolts in a single pass room alternate between four bolts and five bolts across the width of the tunnel, resulting in the staggered pattern shown in Figure 4.1. A single pass room of 61-m in length using this pattern requires 240 bolts in the mine back. Even with each bolt being installed in approximately one minute, installing bolts results in long stretches of time spent without the boring machines running.

It is the use of separate mining machines in anomalous ground conditions that results in the greatest amounts of down time. Changing between machines requires both the movement of the machines, and the pulling down and rehanging of brattice. Especially in single pass areas of the mine, where mining machines must be moved, and brattice be pulled down and reinstalled, over greater distances, much additional time is spent while not advancing the tunnel; in some cases the cut and bolt procedure results in entries taking three times longer as will be shown later. There is also a lot of additional time for installing the bolts themselves, further reducing the productivity. Encountering anomalies greatly reduces the productivity of underground potash mines.

## 4.2 Mining and Bolting Procedures

The cut and bolt procedure used by the potash mines in Saskatchewan is much slower than the normal cutting procedure. This is done to ensure that only acceptable lengths of the entry are left unbolted. To reduce the length of the tunnel that is left unsupported, the boring machines are only allowed to advance so that the operator's area is on the edge of the unsupported ground before stopping and allowing the tunnel to be bolted up to the face. Bolting up to the face requires that the boring machine, and the extensible conveyor system, be backed up to allow access for a separate bolting machine to install bolts in this area. Additional machine movement in the anomalous ground bolting procedure increases the time to mine through anomalous ground.

In normal ground conditions, the mining procedure is very simple. When an entry of a specified length is to be cut, the steps for cutting are as follows:

1. the borer first cuts the entire length of the first pass along the right side of the planned excavation, approximately 60 meters
2. The borer is moved back and prepared for second pass, along the left side of the planned excavation, and finally
3. the second pass is excavated the same distance.

This procedure does not result in very much time spent moving machines around because only one machine is required. Installing bolts within this procedure is however not very fast because as first pass is cut the borer would have to be stopped and backed out periodically to allow the bolting machine to the end of the tunnel. For this reason, a specific anomalous ground procedure has been adapted from the normal ground procedure.

The cut and bolt procedure used by PotashCorp is different in a two-pass room than for a single pass room. Once an anomaly in encountered in a two-pass room:

1. The boring machine is advanced up to the front of the operator's canopy at the back of the machine on the right side of the planned entry.
2. If the anomaly is no longer present, the machine is allowed to continue mining normally, otherwise the borer must stop cutting.
3. The second pass of the room is cut the same length as the first pass along the left side of the planned entry.
4. The borer is backed up approximately sixteen meters to allow the bolting machine access to the mine back near the mine face.
5. The bolting machine is used to bolt the entire width of the mine entry up to 1.5 meters from the mine face, allowing enough space for the borer to expand into the full height of the entry without the bolt heads interfering.
6. The boring machine is advanced to the mine face and cuts the first pass to the point where the canopy is at the last row of bolts.

This sequence is repeated until the anomaly is no longer present or the entry is finished. Both procedures are depicted in Figure 4.3 for a single pass room, and Figure 4.4 for a two-pass width room.

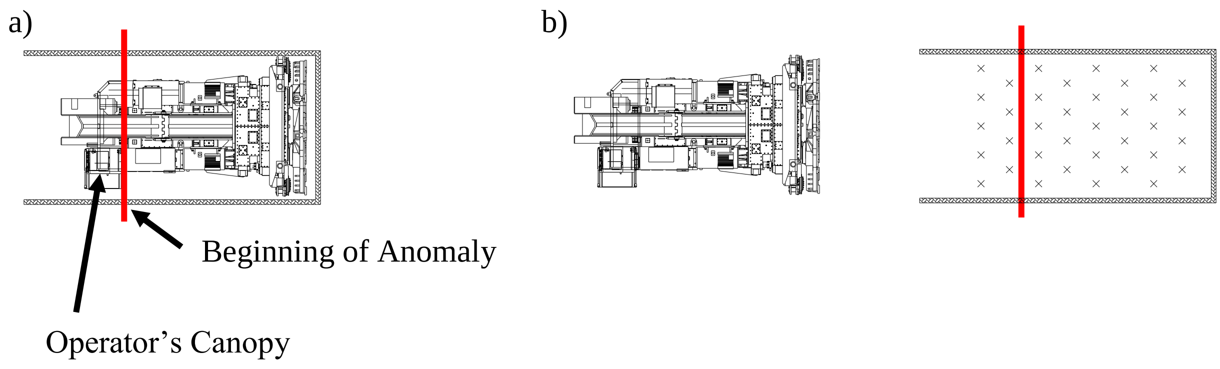



*Figure 4.3 - The current cut and bolt procedure shown in a single pass room. The sequence for cutting is to allow the borer into the entry until the point where the front of the operator’s canopy is even with the start of the anomaly at 7.6 meters (a), the borer is then removed from the entry entirely and bolts are installed (b).*

The procedure is changed for single-pass entries due to the width of the entry mined. Once again, the boring machine is advanced down the entry until an anomaly is located at the front edge of the machine canopy. However, because the room is only a single pass wide, the machine is not lined up for the second pass and is rather simply backed all the way out of the entry. The bolting machine then does the same task as in a two-pass room and the process is continued. The process is slower in a single pass room because the boring and bolting machines each must be backed entirely out of the entry while the other machine is working due to room width constraints. The borer has a maximum tram speed of approximately six meters per minute and the entries currently are driven up to 61 meters, with plans to extend this distance in the future.

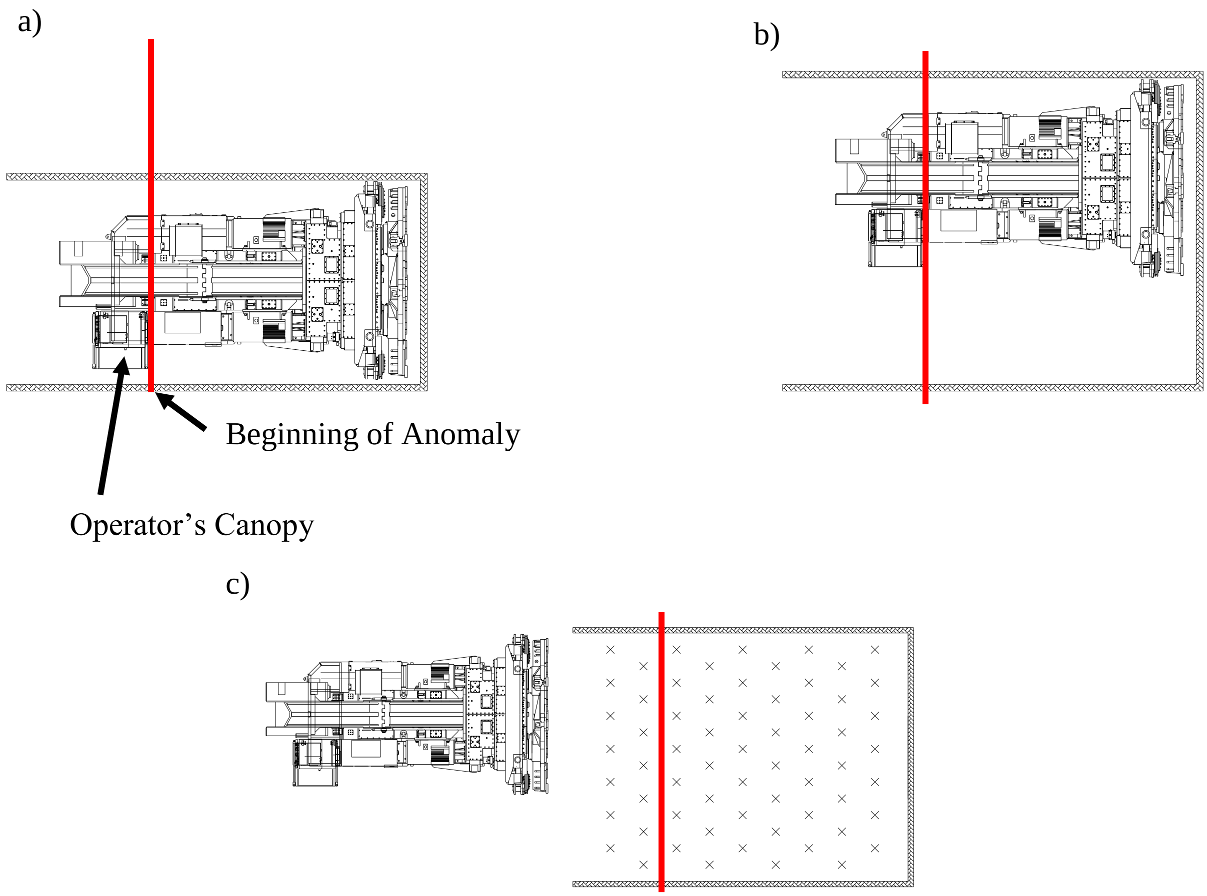



*Figure 4.4 - The current cut and bolt procedure shown for a two-pass room. The sequence for cutting is to allow the borer into the entry until the point where the front of the operator’s canopy is even with the start of the anomaly in first pass at 7.6 meters (a), the borer is moved backward and allowed to cut the second pass to the same depth (b), and finally the borer is backed out far enough to allow the bolting machine access and bolts are installed (c). The borer is not required to move as far back in a two-pass width room as it is required to in a single pass room.*

The anomalous ground cut and bolt procedure is much slower than when mining using the normal ground procedure. Additional time is used for installing bolts and moving the machines in the anomalous ground bolting procedure. The time spent cutting the entry remains the same between the two scenarios. In two-pass width rooms, the average mining speed is approximately constant regardless of the final length of the entry because the mining process is a repeated cycle with identical driving distances in each cycle. On the contrary, the single pass room anomalous ground bolting procedure slows as the tunnel is advanced due to the extended distance that both machines must be advanced and backed up. The machines are backed up approximately sixteen meters in a two-pass room when the machines need to be changed while in a single pass room the machines are backed up the length of the entry, sometimes sixty meters or more. When single pass rooms encounter anomalies, the mine may cut the room two passes wide to avoid the extended machine movements.

### 4.2.1 Cutting Procedures Analyzed

There were several cutting procedures determined to be of interest that were subsequently analyzed. Firstly, the current procedures for both normal and anomalous ground as previously described. For proposed bolting methods, the first proposed machine capability was to be able to bolt the full width of the borer on the fly while mining. This machine would be theoretically ideal for time reasons. A simplification of this machine where only the right side of the conveyor system would be bolted was also analyzed. A further simplification was only installing bolts while the machine is paused, either one at a time or multiple bolts at a time, and this was also analyzed. All options follow the current method of cutting along the right side of the planned entry first, and subsequently cutting the left side of the entry if the room is more than a single pass wide. These methods presented a good breadth of procedural options that could be used to determine the feasible amount of time that could be eliminated from the anomalous ground bolting procedure and the effect of various machine capability options.

#### *4.2.1.1 Bolt While Mining – Full Width of the Boring machine*

The first proposed bolting method to reduce the additional time taken by the current anomalous ground bolting procedure was to bolt the entire width of the entry on the fly while mining. This procedure would use an identical procedure to the normal ground mining procedure but would install bolts while the borer was advancing. The installation of bolts would not slow down the process as both boring and bolting could happen at the same time. In the first pass of a two-pass room, this system would still however only install bolts in the first three meters from the right wall. Any bolts installed further than this would be subsequently cut off by the borer in second pass due to overlap between the two passes. This is the ideal method for installing bolts with respect to time because no additional time is needed to install bolts compared to the normal ground procedure.

#### *4.2.1.2 Bolt While Mining – Right Side of Conveyor Only*

A simpler to implement bolting method where only the right side of the conveyor was bolted on the fly by a borer-mounted bolter was subsequently analyzed. The right side of the machine's conveyor system was chosen because this is the only location that mining personnel are located during the mining sequence, and therefore still provides bolts above the areas personnel are at all times. This machine would operate under exactly the same procedure as a machine for bolting the full width in all cases. Once the borer finished the entry, however, it would be required to use a specialized bolting machine to install bolts in the left-most three meters of the entry as this area would not be bolted while mining. Because of the location of the single pass rooms in PotashCorp mines, the final step of installing bolts in the left most three meters in single pass rooms would not be required because upon the borer being removed from the entry it is cordoned off and no personnel are allowed in. In such an area, this style of machine would result in the same time savings as bolting the full width of the borer.

#### *4.2.1.3 Bolt While Paused – Single Bolt at a Time on the Right Side of the Conveyor Only*

Further simplification of the implementation is possible if the borer is only able to install bolts on the right side of the conveyor when mining is paused. Pausing to install the bolts will reduce the effectiveness of such a system at reducing the time required in an anomalous ground condition. Time is further affected in this case by how many bolts can be installed at once, as this dictates how long mining must be paused. This process, by installing one bolt at a time, results in the boring machine being paused to install bolts the same amount of time as the current cut and bolt procedure and only reduces the amount of time spent moving boring machines around.

#### *4.2.1.4 Bolt While Paused – Two Bolts at a Time on the Right Side of the Conveyor Only*

Similar to the previous procedure, it was also analyzed how the time is affected if multiple bolts can be installed at once while mining is paused. If two bolts can be installed in first pass and two more in second, the time spent waiting for the bolts to be installed can be cut in half by installing both bolts at the same time. If only one bolt can be installed due to space constraints, then only a single bolt is installed at a time and this process results in the same time distribution as the previous method.

Of the proposed methods analyzed, it is generally easy to determine the rank that each method will have in improving time utilization. The ideal solution from a time standpoint is bolting the full width of the entry while mining. This method would eliminate entirely the additional time required to install bolts compared to the normal ground procedure. Some slightly reduced time savings are available if only the right side of the machine can be bolted while in motion. Further reductions in saved time will be achieved if the borer must be paused to install the bolts. Some of that saved time can be recovered if the machine can install two bolts at a time rather than just one at a time. All the proposed methods will reduce the need for the boring machine and a dedicated bolting machine to be moved around and therefore will result in some saved time. The bolting patterns are shown in Figure 4.5 for the proposed full width bolting pattern, and Figure 4.6 for the proposed bolting method for the right side of the conveyor only.

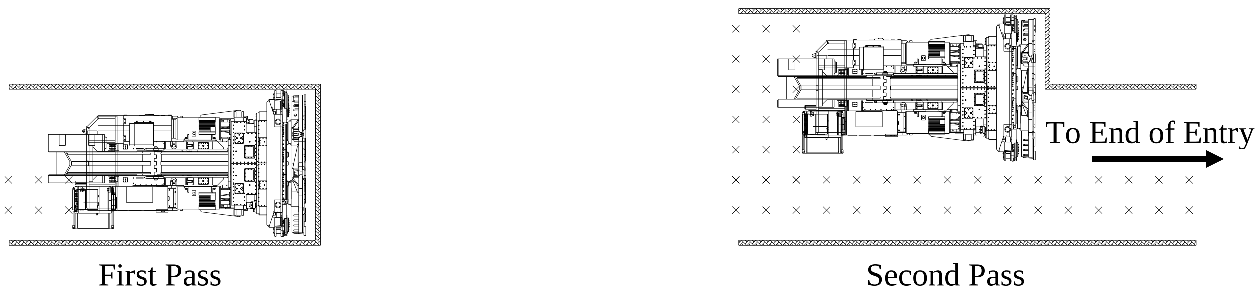



*Figure 4.5 - Proposed bolting procedure for full width of the boring machine. 'X's represent bolt installation locations. This installation pattern is used for 'Bolt while Mining – Full Width of Boring Machine'. Bolts can only be installed in the area outside the overlap of the machine between the two passes, the half of the entry along the right pillar.*

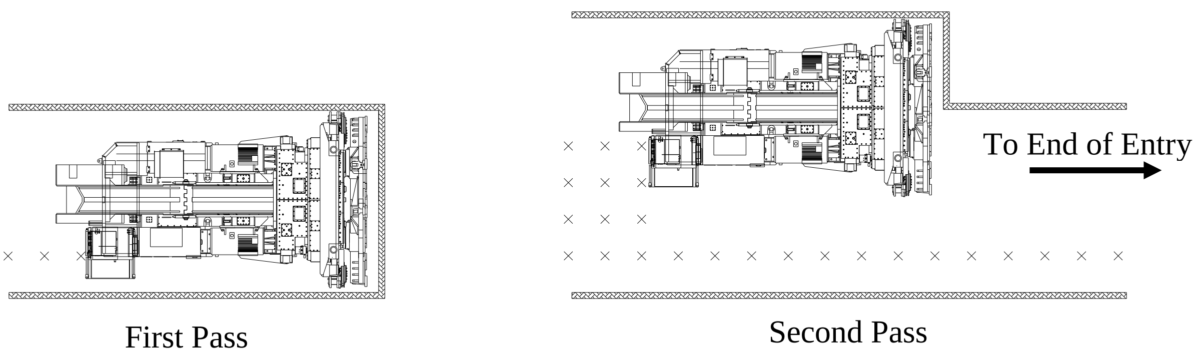



*Figure 4.6 - Proposed bolting method on right side of conveyor only. 'X's represent analyzed bolt installation locations. The same pattern is used for the 'Bolt while Mining – Right Side of Conveyor Only', 'Bolt while Paused – Single Bolt on Right Side of Conveyor Only', and 'Bolt while Paused – Two Bolts at a Time on Right Side of Conveyor Only' analyzed procedures.*

## 4.3 Research Methods

The objective of this sub-study was to determine the potential for time savings by altering the anomalous ground bolting procedure to one utilizing a borer-mounted bolter. Determining the overall expected change in time required several factors to be identified, namely the rates at which individual tasks were completed, the exact cutting procedures to analyze and compare, and the analysis method for both calculation and uncertainty. Individual tasks were determined and timed while observing the mining process underground. With this information, an informed decision was made about what anomalous ground bolting procedure to propose and compare to current processes. Finally, an analysis method needed to be found that allowed both the expected values as well as the standard errors to be calculated. The distribution of time used in each of the various proposed processes, as well as time saved, showed where the majority of time savings are available and how much time can be saved by changing the current anomalous ground bolting procedure.

In the following subsections, the analysis methods used for both the raw calculation and the uncertainty are explained based on the proposed methods outlined earlier.

### 4.3.1 Time Model Used

A single time model was created with the variables necessary, so the same model could be used for all the bolting methods analyzed. The bolting methods are generally very similar, differing only in the length of pause required to install a bolt, the maximum allowable excavation distance before filling in the bolting pattern, and the coverage of the tunnel after the borer-bolter has installed all bolts possible. The time model takes advantage of the fact that the mining process is very repetitive to find the percentage of time used on any specific component of the procedure; the current anomalous ground bolting procedure repeats every eight meters. The percentages stay constant for almost all the methods and scenarios and therefore are all that is required. The only exception is that of a single pass-width room in the current anomalous ground bolting procedure. This procedure requires longer distances to be traversed by the machines as the tunnel gets longer and, as such, requires that the whole of the length of the tunnel be calculated. The model created shows the differences in expected mining rate between the current and proposed options.

The bolting procedure for both the current and proposed bolting patterns are generally similar enough to use one model capable of describing them all. All the procedures start by excavating the first pass to a certain length. Bolts are installed across up to two meters from the right pillar as per the bolting method capability. The borer is then reversed out of the entry and moved sideways to prepare for the second pass if the room utilizes a second pass. The second pass is subsequently advanced the same distance as the first pass, once again installing bolts per the bolting method capability. Finally, the boring machine is removed from the entry and any required remaining area is bolted using a dedicated bolting machine. This process is repeated until the final length of the entry is reached.

Where the current and proposed bolting methods differ is in the allowable length of entry before bolting, the number of bolts that can be installed while mining, and the down time required to install bolts from the miner. Current bolting procedures allow only 7.6 meters of the entry to be excavated before bolting, a limitation done away with using any of the proposed methods. The proposed methods differ from each other in the number of bolts that can be installed while mining and subsequently in the number of bolts required to be installed after the entry is finished. Current bolting procedures do not install any bolts while mining and therefore must install bolts across the full width of the entry at the end of each cutting cycle. Lastly, the amount of time the borer is paused to install bolts from the miner varies between zero, in the case of machines that bolt while mining, and the current bolt installation time to install three bolts, in the case of a system designed to install a single bolt at a time while paused in second pass. Descriptions of some of the variables are shown below in Figure 4.7.

The general equation for the time used is shown below:

$$t = t_{mining} + t_{bolting} + t_{BorerMove} + t_{BolterMove} + t_{Brat} \tag{4.1}$$

where $t$ is the total time,

$t_{mining}$ is the time spent mining,

$t_{bolting}$ is the time spent installing bolts,

$t_{BorerMove}$ is the time spent moving the borer without mining,

$t_{BolterMove}$ is the time spent moving the bolting machine, and

$t_{Brat}$ is the time spent handling brattice.

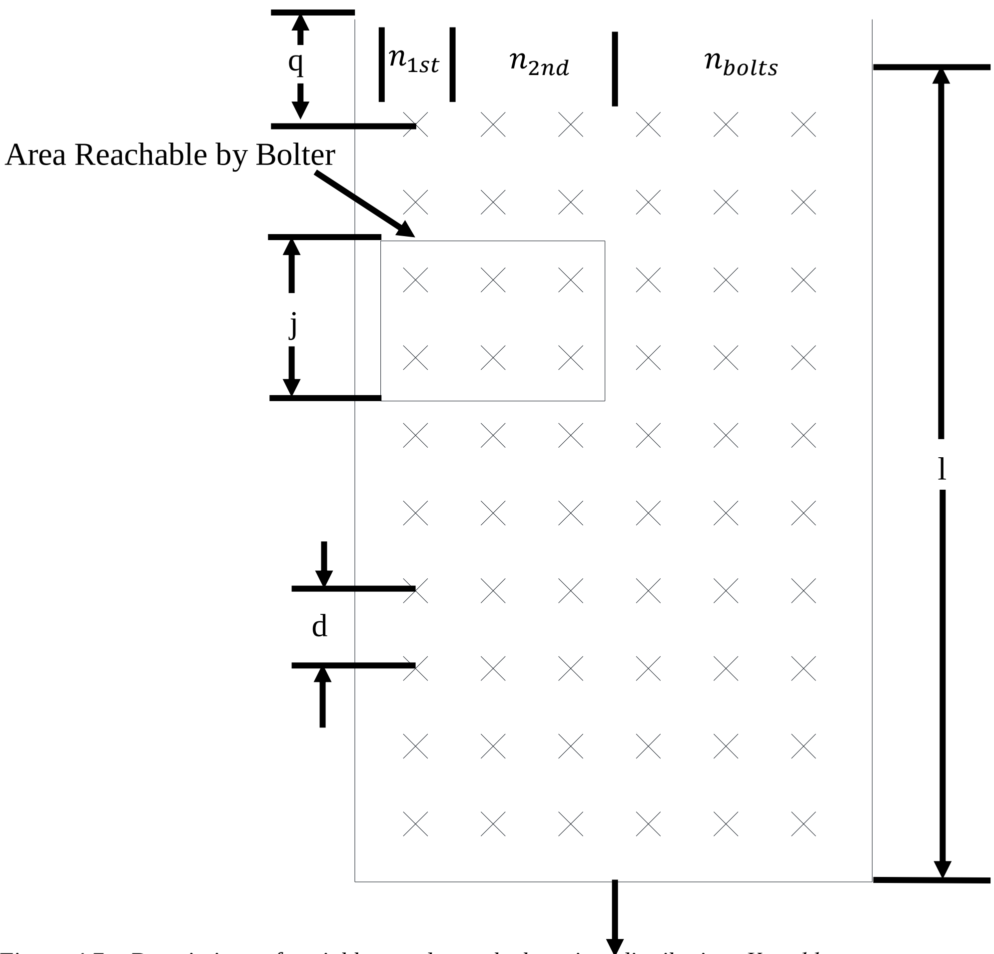



*Figure 4.7 - Descriptions of variables used to calculate time distribution. Variables names are as described in the text. 'X's represent bolt installation locations.*

Note that $t_{BolterMove}$ and $t_{bolting}$ are both equal to zero for a bolting method capable of bolting the full tunnel width. Each of the time components can then be found individually via the following relations:

$$t_{mining} = \frac{n_{pass} l}{a} \tag{4.2}$$

| | | |
|---|---|---|
| where | $a$ | is the mining advance rate (m/hr), |
| | $l$ | is the length of the cutting cycle (m), and |
| | $n_{pass}$ | is the number of passes in the finished entry; |

$$t_{bolting} = \frac{l}{d}\left(e(n_{bolts} + n_{1st} + n_{2nd}) + \frac{n_{pass} k}{j}\right) \tag{4.3}$$

where $n_{bolts}$ is the number of bolts required to finish the width of the entry,

$e$ is the individual bolt installation time (hr),

$d$ is the distance between rows of bolts in the direction of advancement (m),

$k$ is the time taken to move bolter from one bolting position to another (hr),

$j$ is the number of bolt rows reachable by the bolter at once,

$n_{1st}$ is the number of bolt install cycles per first pass pause, and

$n_{2nd}$ is the number of bolt install cycles per pause during second pass;

$$e = s + \frac{u}{v} \tag{4.4}$$

where $s$ is the drilling cycle time (m/hr),

$u$ is the time to change the drill bit (hr), and

$v$ is the number of bolts between changing drill bits;

$$t_{BorerMove} = f + \frac{\left(n_{pass} + 1\right)l}{b} + 2\left(n_{pass} - 1\right)c + \frac{gln_{pass}}{p} \tag{4.5}$$

where $f$ is the time required to change machines (hr),

$b$ is the movement rate of borer when not mining (m/hr),

$p$ is the distance mined before changing bits (m),

$c$ is the time delay to move sideways from first to second pass (hr), and

$g$ is the time required to change borer bits (hr);

$$t_{BolterMove} = f + \frac{n_{pass}(l + q)}{m} \tag{4.6}$$

where $m$ is the bolter movement rate (m/hr), and

$q$ is the extra distance left to allow free movement of the bolting machine (m);

$$t_{Brat} = \frac{l}{o} + \frac{l}{r} \tag{4.7}$$

where $o$ is the brattice hang rate (m/hr),

$l$ is the length of the cutting cycle (m), and

$r$ is the brattice removal rate (m/hr).

For a single pass room using the current anomalous ground bolting procedure, some of the terms are changed. This is due to the increased traversal distances as the length of the drift increases and therefore only affect machine movement and brattice handling times. This results in the following altered formulas:

$$t_{BorerMove} = \frac{gln_{pass}}{p} + \sum_{i=1}^{n_{cut}} f + \frac{2il}{b} \tag{4.8}$$

$$t_{BolterMove} = \sum_{i=1}^{n_{cut}} \left(f + \frac{il+q}{m}\right) \tag{4.9}$$

$$t_{Brat} = \sum_{i=1}^{n_{cut}} \left(i\left(\frac{l}{o} + \frac{l}{r}\right)\right) \tag{4.10}$$

where $n_{cut}$ is the number of cutting cycles required for a completed entry.

It can be seen that if only one cutting cycle was used to complete the entry, or the whole entry was cut at once, the above formulas simplify to those shown earlier in equations 4.5 through 4.7. The values used for each of the method specific parameters, $n_{1st}$, $n_{2nd}$,$n_{bolts}$ and $l$, are shown below in Table 4.1, for a two-pass room, and Table 4.2, for a single pass room. These values are determined via geometric constraints.

*Table 4.1 - The parameter values for a two-pass room. The numbers of bolt cycles used represent the number of bolts deemed possible to install in each individual pass given geometric constraints. Note that zero only indicates that no additional time is used as the process is either not done or done while mining. Using a 1.2-meter bolt spacing, only one bolt installable in first pass and three additional bolts in second pass. Under an altered, compressed bolting pattern, additional bolts could be possible. Installing one bolt in first pass and three more in the second pass is not ideal for installing multiple bolts at a time, which would save a maximal amount of time installing two bolts in each of the first two passes.*

| | **Base Case** | **Bolt while Mining - Full Width** | **Bolt while Mining - Right Side** | **Bolt while Paused - Single** | **Bolt while Paused - Multiple** |
|---|---|---|---|---|---|
| $n_{1st}$ | 0 | 0 | 0 | 1 | 1 |
| $n_{2nd}$ | 0 | 0 | 0 | 3 | 2 |
| $n_{bolts}$ | 7 | 0 | 3 | 3 | 3 |
| $l$ | 7.6 m | 61 m | 61 m | 61 m | 61 m |

*Table 4.2 - The parameter values for a single pass room. Note that zero only indicates that no additional time is used as the process is either not done or done while mining. Here it can be seen that there are only three unique outcomes, the current anomalous ground bolting procedure, a proposed bolting while mining procedure, and a proposed bolting while paused procedure. Again, if two bolts are installed in first pass, installing multiple bolts at a time would be more effective than installing a single bolt at a time.*

| | **Base Case** | **Bolt while Mining - Full Width** | **Bolt while Mining - Right Side** | **Bolt while Paused - Single** | **Bolt while Paused - Multiple** |
|---|---|---|---|---|---|
| $\boldsymbol{n_{1st}}$ | 0 | 0 | 0 | 1 | 1 |
| $\boldsymbol{n_{2nd}}$ | 0 | 0 | 0 | 0 | 0 |
| $\boldsymbol{n_{bolts}}$ | 4 | 0 | 0 | 0 | 0 |
| $\boldsymbol{l}$ | 7.6 m | 61 m | 61 m | 61 m | 61 m |

The time model developed for this study can model the time consumption and distribution of all the various methods and scenarios analyzed by this study. Some obvious outcomes can already be seen, such as the rank of the proposed bolting scenarios in terms of saved time. Bolting the full width of the boring machine while mining is the quickest way to traverse anomalous ground while installing a single bolt at a time while paused will be the slowest. There is also indication that installing multiple bolts at a time may not gain critical time reductions; the number of installable bolts in each pass do not line up well with installing two bolts at a time. These equations allow for the calculation of the expected time distribution and total time reduction possibilities.

### 4.3.2 Rate Determination

Rate determination was done by observing the mining process over the course of a month. Based on observations, it was noted that the time taken to install bolts and the time spent boring the tunnel amounted to the largest amounts of time used within the anomalous ground bolting procedure. The sheer number of bolts installed caused the bolt installation time to take more time than was spent boring the tunnel in some cases. The boring machines additionally move slower while cutting than otherwise. Machine movement and brattice installation time was comparatively less. Effort was then focused on determining the bolt installation and tunnel boring rates to a higher level of accuracy than other less influential parameters.

#### *4.3.2.1 Bolt Installation Time*

One of the largest areas of increased time usage in anomalous ground conditions is that of installing bolts. The current process for in anomalous ground conditions requires that mining be stopped while the bolts are installed because the bolting machine must have unrestricted access to the mine back. Data was collected over the course of two days spent underground with a video camera. It is important that accurate averages be obtained due to the large portion of time that this process consumes in the anomalous ground bolting procedure. To get a sufficient number of data points to visualize the distribution of times from limited sample sizes, a bootstrap resampling technique was used to create estimated data points.

To maximize the data from a limited time underground, the data was split into different processes. Cycles which then deviated from the normal cutting cycle for other intermittent reasons, such as the changing of drilling bits, did not have to be removed from the data set. There were seven processes identified: the hole drilling time, the time to retract the drill bit, the time to rotate from the drill bit to the bolt insertion drill, the time to push the bolt into the hole, torquing time, retracting the bolt head and loading a new bolt, and lining up the next bolt location. The number of points collected for each of the various categories is shown in Table 4.3. The data was collected over several days using a video camera. Two operators were observed, with very similar results, to be more representative of the overall time average. It is not expected that observation of the process had any effect on the data. Data points collected are presented in Appendix A. The final cycle time for the bolting time is equal to the sum of the averages of each of the processes measured.

For later simulation purposes, the data was fitted more accurately using Monte Carlo methods. The distribution was first made more visible by recombining the data points from the original data set. For the number of available points shown in Table 4.3, the total number of possible combinations is greater than thirty million. A portion of these points, 150 000, were randomly selected with replacement and used to create a histogram of the predicted time distribution, shown in Figure 4.8. When the simulation was rerun with the same number of points multiple times, it was shown by inspection that the distribution stayed stable to three decimal places. Even though the number of samples is small, the resulting total standard deviation is low, and therefore Monte Carlo methods can be used.

*Table 4.3 - Number of Data Points Used in Monte Carlo Resampling Technique. The standard error values as a percentage of the mean for some processes are high. These processes however represent small means, e.g. 'Retract bolt head and load next bolt' has an average duration of 5 seconds, and therefore do not affect the overall average very greatly. The overall standard error for the bolting procedure is 1.7% of the average value.*

| Process | Number of Data Points | Standard Error as Percentage of Mean |
|---|---|---|
| **Bolt hole drilling time** | 12 | 2.2% |
| **Drill bit retract time** | 13 | 5.7% |
| **Drill head rotation** | 13 | 4.1% |
| **Push bolt into hole** | 13 | 6.8% |
| **Tighten bolt** | 13 | 4.1% |
| **Retract bolt head and load next bolt** | 11 | 8.5% |
| **Lining up next hole** | 8 | 3.5% |
| **Total Procedure** | - | 1.7% |

Resampling allows the fit of the normal distribution to the data to be analyzed. As shown in Figure 4.8, the distribution of the data is relatively normal with a heavy right tail; the skew of the data can be calculated to be 0.2975. Determining the fit of the distribution was done using the root sum squared error method. While generally not used for continuous distributions because of inaccuracies, the method can be used here because the underlying data can only take whole integer values. If the root sum square error is taken at each possible point the method renders accurate results. The mean and standard deviation of the simulation were 71.2 and 3.79 seconds, respectively. The calculated mean root sum squared error between the generated distribution and a normal distribution was found to be 759 points per one second bin while the maximum one second bin count was 15 735 points. While reasonably accurate, improved accuracy can be achieved using a different distribution that allows skew.

By visually comparing several other distributions to the simulated resuls, it was found that the generalized extreme value distribution was a better fit to the data than the normal distribution. The distribution allows for the fitting of the heavy right tail pictured in Figure 4.8 by using three parameters: $k$, the shape factor; $\mu$, the location parameter; and $\sigma$, the scale parameter. The best fitting distribution was found to be at $k$ = -0.187, $\mu$ = 69.7 seconds, and $\sigma$ = 3.56 seconds respectively. The root mean squared error for the distributions fit was found to be much lower than the value for the normal distribution, 246 simulated points per bin, equivalent to 1.57% of the maximum bin count in the probability distribution.

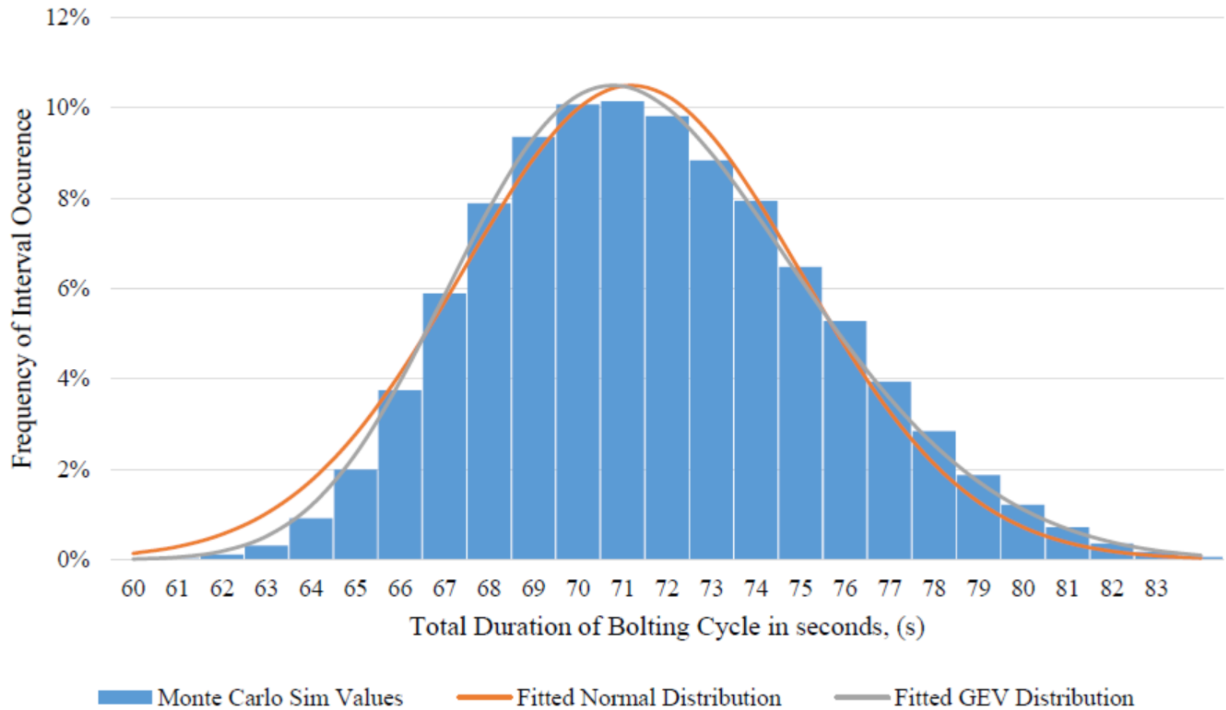



*Figure 4.8 - The histogram of the results of the Monte Carlo simulation of bolting time. The right tail of the distribution is heavier than the left tail, indicating that the assumption of a normal distribution may be poor.*

#### *4.3.2.2 Borer Advance Rate*

Internal data from PotashCorp made the advance rate of the borer while cutting simpler to calculate [36]. A current study at the Lanigan division mine site has found the mean mass of ore per cutting hour realized by the relevant boring machines, as well as the amount of potash ore realized per meter. These two figures allow for the mining rate to be calculated via the following relation:

$$v = \frac{\dot{m}}{\rho} \tag{4.11}$$

| | | |
|---|---|---|
| where | $\dot{m}$ | is the average mass flow rate from the borer (kg/hr), |
| | $\rho$ | is the linear density of the potash for the borer (kg/m), and |
| | $v$ | is the advance rate of the borer (m/hr). |

The standard error of the mean cannot be found using statistical variance combination formulas due to the division involved. A calculus based formula can be used to find the standard error of the advance rate as shown:

$$\delta(v) = \sqrt{\left(\frac{\delta(\dot{m})}{\rho}\right)^2 + \left(\frac{\dot{m}\delta(\rho)}{\rho^2}\right)^2} \tag{4.12}$$

where $\delta(x)$ is the standard error of variable $x$.

This results in a mean advance rate of $0.29 \pm 0.0071$ m/min.

*4.3.2.3 Other Parameters*

Many other parameters were required to calculate the distribution of time and the expected time savings of each proposed method. Preliminary information, determined from time spent underground with a video camera, for these values was used to calculate the overall time distribution. It was found that even with very high standard errors, the overall 95% confidence interval for each of the procedures analyzed was within 10% of the mean value. It was further determined that the uncertainty in the average bolting time was the largest contribution to the uncertainty. Due to the lack of effect on the overall procedural time values, it was determined that it was not worth obtaining more accurate input values for the expected marginal effect on the final results. Table 4.4 below show the values used for calculation and the associated values and standard errors.

*Table 4.4 - Extra parameters required to calculate the overall time distribution of each individual method. Many of the constants were estimated from limited time spent observing the processes and therefore have high standard errors associated. It was found that the end results were not very sensitive to standard errors of the magnitude shown so it was decided not to persue more precise data.*

| Related Machine | Task | Time Model Parameter | Expected Time | Estimated Standard Error |
|---|---|---|---|---|
| **Borer Movements** | Advancing/Backing up (Not Cutting) | $b$ | 4.6 m/min | 1.5 m/min |
| | Moving Sideways ($1^{st}$ to $2^{nd}$ pass) | $c$ | 2.0 min | 0.2 min |
| | Change Bits | $g$ | 5.0 min | 4.0 min |
| | Distance between changing bits | $p$ | 122 m | 0 m |
| **Brattice Handling** | Hang Brattice | $o$ | 3.0 m/min | 0.3 m/min |
| | Tear Down Brattice | $r$ | 30.4 m/min | 4.6 m/min |
| **Bolting Machine** | Advance Bolter | $m$ | 83.2 m/min | 15.2 m/min |
| | Set up new machine | $f$ | 15.0 min | 4.0 min |
| | Change bits in bolter | $u$ | 0.5 min | 0.3 min |
| | Bolts between changing bits | $v$ | 27.5 bolts | 2.5 bolts |

### 4.3.3 Uncertainty Analysis Methods Used

Finding the uncertainty associated with each of the expected time values was more challenging than finding the expected value. Expected values were found using the proposed procedure and the rates found for each of the individual tasks. Assuming normality of the data allows standard statistical formulas for the variance to be used to propagate the uncertainty of the data. Unfortunately, however, there are times where two variables with associated uncertainties must be divided and variances cannot be accurately combined in this scenario. To propagate the standard deviations of the mean through such a calculation, a calculus based error calculation was used. It was found that if the deviation was sufficiently small compared to the expected value that the calculus method resulted in very comparable results to using full variance combination equations. Once the standard deviation of the mean had been propagated using a calculus based equation it was assumed that the data was again normal and variances were once again used.

Further analysis of this assumption is examined using Monte Carlo simulation techniques at a later point in this thesis.

## 4.4 Results

The results found from the time analysis show the difference in expected procedural time for the current normal ground procedure, current anomalous ground bolting procedure, and each proposed bolting method, as previously outlined, in both single and two-pass entries. It can be shown that the maximum time saving available for any method is the difference between the current normal ground procedure where no bolts are installed and the current anomalous ground bolting procedure. Every one of the proposed procedures results in a portion of this available saved time being realized.

### 4.4.1 Current Procedure Time Results

Study data shows that the current time distribution in salt anomalies for the PotashCorp mines contains considerable amounts of non-cutting time. The current cutting time percentage in the anomalous ground bolting procedure is 34% and 26% for the two-pass and single pass rooms, respectively. In the two-pass room, shown in Figure 4.9, large amounts of time are spent moving both the boring and bolting machines as well as installing bolts. Brattice does not consume noticeable time in the two-pass room anomalous ground bolting procedure. For the single pass room, shown in Figure 4.10, increased time is spent moving the bolting machine and hanging brattice. This is because the brattice and bolting machines must now be moved the entire length of the entry each time there is a machine change required, as opposed to only approximately ten meters with the two-pass room. Additionally, the bolting machine takes a larger amount of time to set up, and retrieve bolts from storage and ready them for installation in a single pass room than in a two-pass room. The borer, by comparison, is less time-intensive to set up and therefore makes less of a difference from a single to two passes.

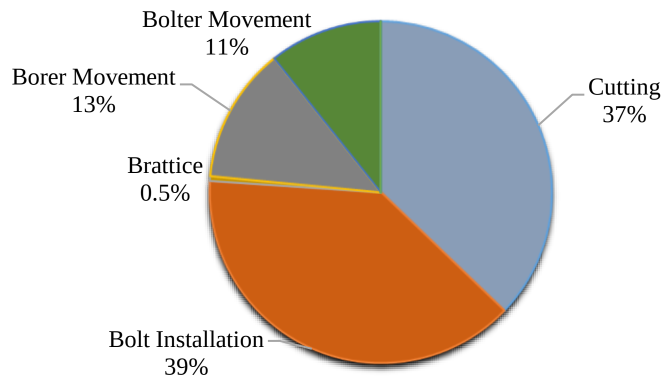



*Figure 4.9 - Two-pass room time distribution using existing procedure in anomalous ground. The largest portion of time is used on the installation of roof bolts.*

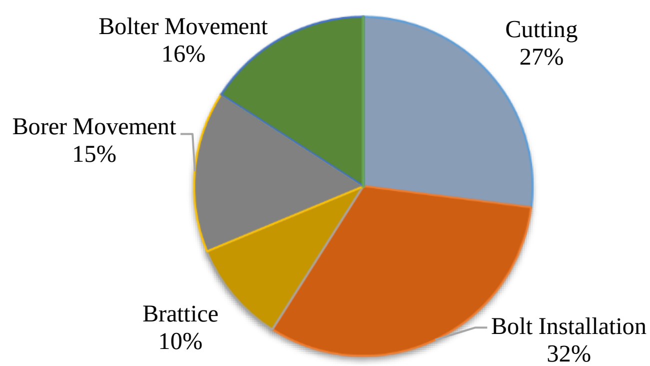



*Figure 4.10 - Single pass room time distribution with existing procedure in anomalous ground. The largest time portion is still used installing roof bolts as it was in a two-pass room. In the single pass room, the time amounts for borer movement, brattice, and bolter movement all increase as the length of the entry increases. These numbers are based on a sixty-one-meter entry.*

Outside of salt anomalies, the distribution of procedural time differs greatly from the distribution of time required in anomalies. In both the two-pass room and the single pass room, cutting accounts for approximately 88% of the total time in the entry. In ground with no salt anomalies, it is not required to install roof bolts, as the stability of the ground is already acceptable, so there is no time spent installing the roof bolts or moving the bolting machine. Borer movement is still required to go from first to second pass in the two-pass room and to start the borer in both rooms. The time breakdown for the two-pass room is shown in Figure 4.11 and is like that of the single pass room, shown in Figure 4.12. The difference in time required between the anomalous and non-anomalous ground conditions shows where the potential for improved production rates can be found.

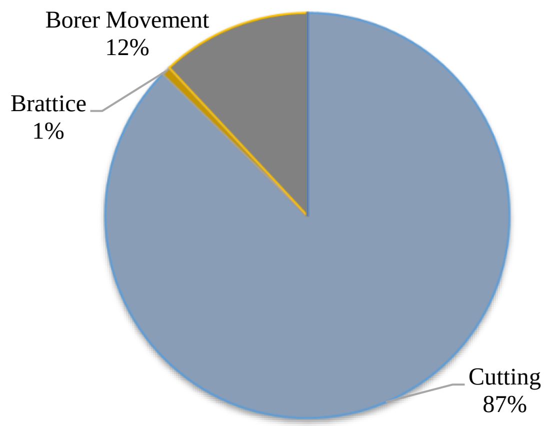



*Figure 4.11 - Two-pass room time distribution outside anomalous ground. Cutting makes up most of the total time in the mined entry. Installation of bolts and the movement of the bolting machine are no longer required because no bolts are installed in normal ground. Borer movement, which includes starting the boring machine, is still required for moving into second pass and in starting the machine.*

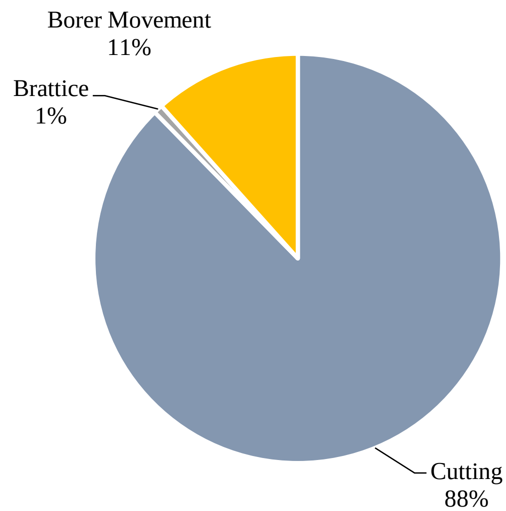



*Figure 4.12 - Single pass room time distribution outside anomalous ground. The results are very similar to those in a two-pass room shown in Figure 4.11.*

The time distributions for the current anomalous ground bolting procedures show room for time savings. If the normal ground condition procedure is taken as the maximum effective advance rate possible through anomalous ground conditions, the existing anomalous ground bolting procedure uses 235% and 325% more than the minimum time required for a two-pass and single pass room respectively. In the two-pass room anomalous ground bolting procedure, shown in Figure 4.11, the largest portion of time is spent installing bolts in the mine back. Minimal time is spent installing brattice because the boring machine can be pulled to one side of the brattice and the bolting machine allowed to drive past on the other side, allowing the brattice to remain in place. Only relatively small additional amounts of time are required to maneuver the borer and bolter. The biggest difference between a two-pass room and a single pass room is the time spent handling brattice. Due to the need to remove and rehang brattice down the entire entry length in a single pass room, as the tunnels get longer so too does the amount of time required for brattice handling. Reducing the need to move brattice in single pass rooms and not stopping to install bolts constitutes most of the time available to save in anomalous ground conditions.

### 4.4.2 Proposed Procedure Results

All the proposed procedures for bolting allow an entire pass to be bored without stopping to change machines. This results in reduced machine movement time. In the case of a system only able to install bolts while mining is paused, the main portion of the saved time is only from the machine movement time categories. Where bolts can be installed during mining, there is an additional savings shown in the time spent installing bolts. Each of the proposed processes greatly reduced the overall time to cut entries in these areas.

Systems designed to only bolt when the borer stops (on pause) are expected to reduce entry cutting time the least of any of the proposed solutions. As can be seen from Table 4.5, the total amount of time spent installing bolts is relatively unchanged from the current procedure if only one bolt is installed at a time while the boring machine is paused. Some bolt installation time is saved if multiple bolts can be installed at a time relative to the current cut and bolt procedure. The time savings for this type of system can mostly be attributed to both borer and bolter movement reductions of $0.20 \pm 0.06$ and $0.25 \pm 0.07$[1] hours for the borer and bolter respectively for every hour spent cutting with the boring machine. In Table 4.6, the time savings in a single pass room are shown. In the single pass case, the systems that bolt on pause are expected to reduce the time spent installing bolts by $0.63 \pm 0.023$ hours per cutting hour in addition to reducing machine movement. Adding the ability to install more than one bolt at once reduces the time spent installing bolts by an additional $0.28 \pm 0.015$ hours per cutting hour in both the single and two-pass room results. It was originally thought that by installing multiple bolts at a time the overall time would be reduced more but this turned out not to be the case. Neither bolt on pause system is shown to

[1] Note: All $\pm$ values are the standard error, approximately a 67% confidence interval

reduce the time spent installing bolts in a two-pass room very much, and therefore, is not as effective as a bolt on the fly system. Additionaly, it can be seen that the time spent in an anomalous ground condition is reduced by using the single pass versions of the bolt while paused systems compared to the two-pass room version. Under the current anomalous ground bolting procedure, widening a single pass room to a two pass room is considered in order to reduce the time spent in these areas. If using a bolt while paused system, the opposite consideration would be relevant based on time spent.

*Table 4.5 - The amount of time taken for each proposed bolting procedure in a two-pass width room. All numbers are in hours and are scaled per 1.00 cutting hours. Each row, excluding the standard error* $\sigma$*, totals 2.69 hours.*

| | Cutting | Bolt Installation | Brattice | Borer Movement | Bolter Movement | Saved Time | $\sigma$ |
|---|---|---|---|---|---|---|---|
| **Base Case** | 1.00 | 1.04 | 0.01 | 0.34 | 0.29 | | 0.11 |
| **Bolt on Pause Right Side** | 1.00 | 1.02 | 0.01 | 0.14 | 0.04 | 0.49 | 0.04 |
| **Multi-bolt Right Side on Pause** | 1.00 | 0.74 | 0.01 | 0.14 | 0.04 | 0.77 | 0.04 |
| **Bolt on the Fly Right Side** | 1.00 | 0.45 | 0.01 | 0.14 | 0.04 | 1.05 | 0.04 |
| **Bolt on Fly Full Machine** | 1.00 | 0.00 | 0.01 | 0.14 | 0.00 | 1.54 | 0.04 |

*Table 4.6 - The time consumed for each proposed bolting procedure in a single pass room. All numbers are in hours and are scaled per 1.00 cutting hours. Each row, excluding the standard error* $\sigma$*, totals 3.97 hours. The bolt installation time can be seen to be reduced by the bolt on pause systems because both systems require a specialized bolting machine to finish off the left side of the bolting pattern after the borer is finished the entry. In a single pass entry however, the entry is cordoned off once the borer is finished and therefore the bolting pattern would remain unfinished.*

| | Cutting | Bolt Installation | Brattice | Borer Movement | Bolter Movement | Saved Time | $\sigma$ |
|---|---|---|---|---|---|---|---|
| **Base Case** | 1.00 | 1.19 | 0.65 | 0.62 | 0.51 | | 0.24 |
| **Bolt on Pause Right Side** | 1.00 | 0.57 | 0.01 | 0.13 | 0.00 | 2.26 | 0.04 |
| **Multi-bolt Right Side on Pause** | 1.00 | 0.28 | 0.01 | 0.13 | 0.00 | 2.54 | 0.04 |
| **Bolt on the Fly Right Side** | 1.00 | 0.00 | 0.01 | 0.13 | 0.00 | 2.83 | 0.04 |
| **Bolt on Fly Full Machine** | 1.00 | 0.00 | 0.01 | 0.13 | 0.00 | 2.83 | 0.04 |

Further time reductions are possible if a machine can be designed for installing bolts while mining is in progress. Mining does not have to stop during the bolting cycle in this case, resulting in no additional time spent installing bolts if the entire width of the entry can be bolted from the boring machine. In the single pass room case, there is no time difference between installing bolts across the width of the machine as opposed to installing only along the right side of the machine. Bolts along the left side of the entry, if not installed while mining, would not be installed at all because do not re-enter a room after cutting is complete. The total saved time in a two-pass room is expected to be $1.05 \pm 0.13$ and $1.54 \pm 0.15$ hours per cutting hour for bolting the right side of the conveyor and bolting the full machine width, respectively. In the single pass room, due to the lack of need to install the bolts down the left side of the entry, the saved time for both proposed machines is $2.83 \pm 0.30$ hours per cutting hour.

The saved time for all cases is considerably higher in single pass rooms for two reasons: There is more potentially saveable time in the single pass room procedure compared the two-pass room procedure, and a reduced number of bolts installed in the single pass room. The current anomalous ground bolting procedure consumes 3.97 hours per cutting hour in a single pass room compared to only 2.69 hours in a two-pass room; much of the additional time is in categories targeted for time savings by proposed bolting methods and therefore presents more potentially saveable time in single pass. This is based on a 61 meter length of entry, with varying lengths to be disscussed later. These rooms also would not require bolts down the left side of the mine room for the reasons outlined previously. Less machine traversal time in addition to the reduced requirements for bolts in many of the bolting procedures are the cause for the greatly reduced time requirements in a single pass entry.

The ranking of the proposed anomalous ground bolting procedures' time savings was as expected. The reduction in required time is shown to be less when the bolting system used requires mining to stop for bolt installation. The slowest system compared was when installing a single bolt at a time with mining paused. Slight additional time reductions were shown when installing multiple bolts at a time and further reductions if the system can install bolts without pausing the boring machine. What was unexpected was the magnitude of the changes in reduced time. Even the slowest system compared was calculated to reduce time by approximately 78% of the maximum time reduction achievable in a single pass room. Although the time saved is shown to be less in a two-pass room, this result shows that even the slowest system analyzed will result in time savings. Another point of interest was the minimal change by which installing multiple bolts at a time had on the overall time reduction. It was assumed that this feature would result in much more considerable time reduction than was shown in the two-pass room. It was also found that the room for time improvement was not as great compared to installing a single bolt at a time in the two-pass room. These results can help to influence better decision making on final machinery design goals.

### 4.4.3 Variation in Time Saved with Different Length Single Pass Rooms in Anomalous Ground

While the total time saved in a two-pass-width room is a constant percentage of the total time currently used, single pass rooms have a varied amount of time saved depending on the length of the entry. In a room with the additional width of a two-pass room, the boring machine and bolting machine can be driven past each other in the tunnel so the machines only have to be backed up far enough to be considered out of the way. Due to the need to remove the boring machine entirely from the entry to install bolts in a single pass room, additional length in the tunnel requires greater driving distances and therefore additional time to complete. Specifically, the boring machine, a very slow-moving machine, can take an extremely long time to advance and back out of a tunnel. Further time is used in removing and reinstalling the ventilation brattice in the entry.

The total time to finish the entry per cutting hour can be found to scale linearly as the length of the entry increases in single pass scenarios using the existing procedure. The slope of the total time required to finish the entry is directly related to the distance the mine face is from the beginning of the tunnel; greater distances require more time for the borer and bolter to traverse the length of the tunnel. The total time spent cutting in the entry is proportional to the total length of tunnel cut. Therefore, when the total time spent in the entry is scaled by the amount of time spent cutting, the following relation is obtained:

$$t = 0.023l + 2.58 \tag{4.1}$$

where $t$ is the total time spent in hours per cutting hour, and

$l$ is the total length of the finished tunnel in meters.

More importantly, the percentage of time spent cutting, $\%\eta$, is equal to the reciprocal of $t$ from equation 4.1:

$$\%\eta = \frac{1}{0.023l + 2.58} \tag{4.2}$$

Both equations are only valid for distances composed of multiple complete cutting cycles. The relationship between the borer utilization rate and the length of the single pass entry is shown in Figure 4.13.

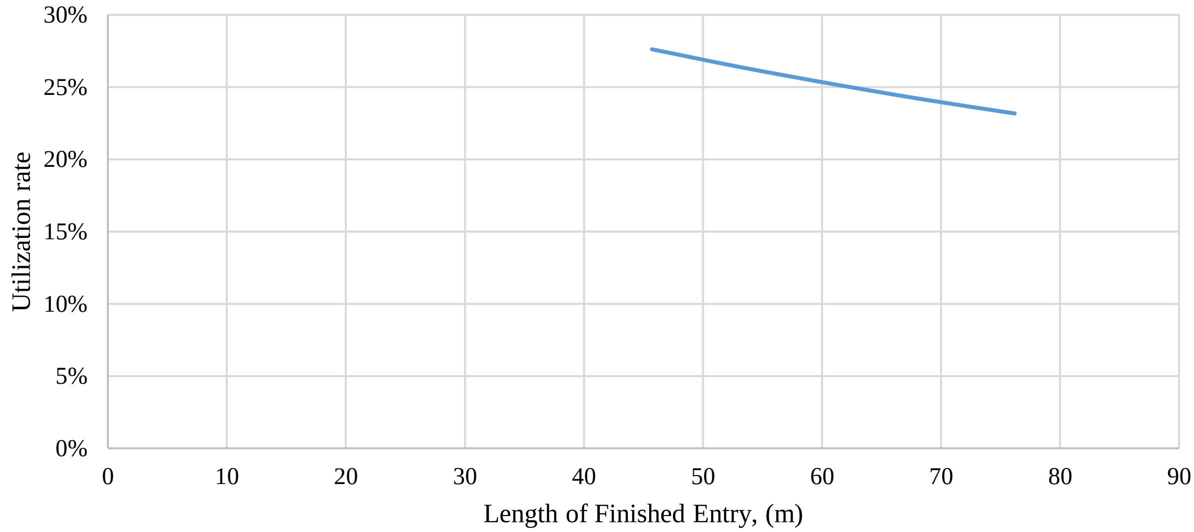



*Figure 4.13 - Utilization rate of the boring machine as entry length increases in a single pass room, following current anomalous ground bolting procedures.*

The total time saved per cutting hour is, likewise, a linear relationship when compared to the length of the finished entry. It can be seen in Figure 4.14 that the four methods analyzed follow parallel trendlines in time saved. The slopes of the lines are identical to the rate that the time required for the current anomalous ground bolting procedure increases. An explanation for this match is that the analyzed proposed bolting sequences are unaffected by the length of the entry in a single pass room because they do not require backing out of the entry multiple times before finishing the entry. The utilization rates for the borers are, therefore, a constant for each of the proposed bolting procedures and are shown in Table 4.7.

*Table 4.7 - The percentage of time spent cutting in an anomalous single pass entry.*

| Cut and Bolt Procedure | Utilization Rate |
|---|---|
| **Bolt on the Fly - Full Machine Width** | 86.0 % |
| **Bolt on the Fly - Right Side Only** | 86.0 % |
| **Bolt while Paused - Single Bolt at a Time** | 57.8 % |
| **Bolt while Paused - Multiple Bolts at a Time** | 69.2 % |

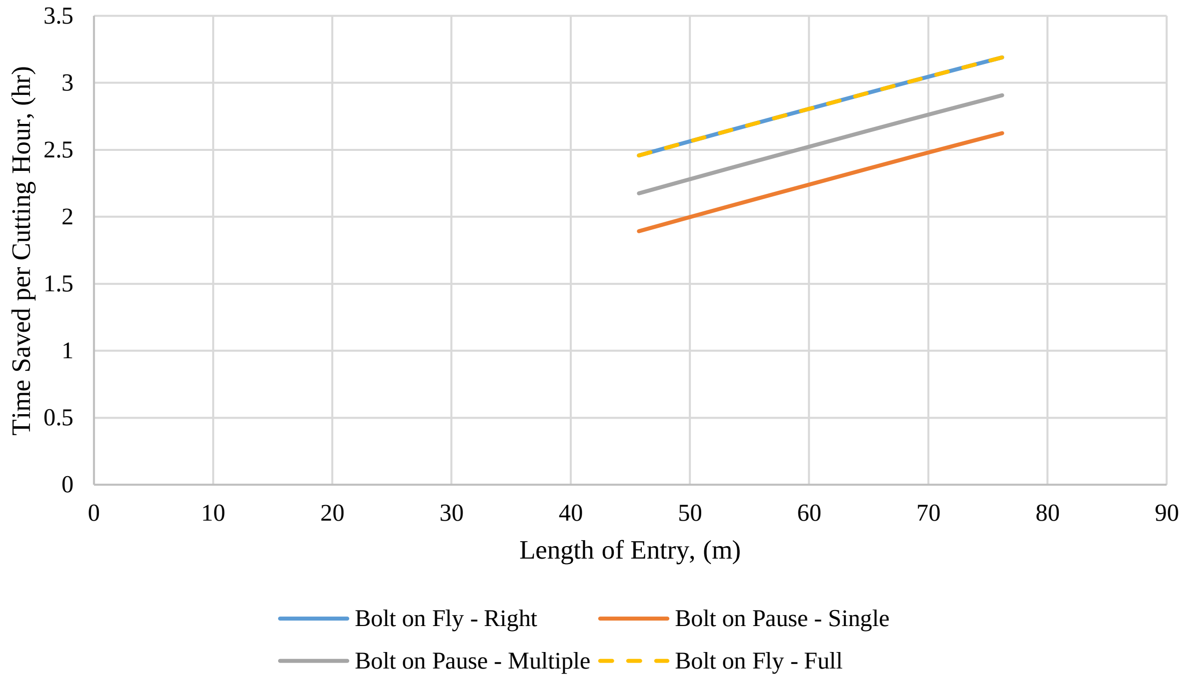



*Figure 4.14 - The amount of time saved per cutting hour for each of the proposed bolting methods in single pass rooms. As discussed previously, the time spent in a single pass entry is unchanged whether the full width or only the right side is bolted when using a bolt on the fly solution, therefore, the time saved for these two scenarios overlaps.*

While all the proposed bolting methods provide an improvement in the mining rate when anomalous ground is encountered anywhere underground, the effect is greatest in single pass rooms. The saved time is further amplified by the length of the single pass entry. Currently, the anomalous ground bolting procedure is affected by the length of the tunnel due to a need to remove the boring machine completely from the entry every eight meters. The total time used by the proposed anomalous ground methods are all unrelated to the full length of the tunnel. For this reason, longer single pass tunnels will see a greater benefit from using an alternative cut and bolt method.

### 4.4.4 Uncertainty Analysis Using the Monte Carlo Method

Uncertainty calculations were also undertaken using a Monte Carlo method. There were three reasons for doing this: first, that there were instances where the division of one random variable by another was required and therefore a calculus based error approximation was used instead of a statistical method; secondly, the Monte Carlo method made it possible to check the normality of the final distribution given normality in the initial random variables; finally, this made it possible to find the standard error between random variables that are not independent, such as the amount of time saved by various procedures. Using a Monte Carlo simulation of the entire calculation allows all points to be analyzed.

It was found that generally the standard deviation values for the total relative time did not vary greatly from one method to the next. The values for each can be seen in Table 4.8. It was found by running the Monte Carlo simulation several times that with 1 000 000 test points, reasonably low variation in the output values were found. The precision level is correlated to the inverse square root of the number of test points so every additional decimal point of accuracy is obtained by a 100-fold increase in the number of test points. The Monte Carlo simulations therefore verified the analytical calculations to reasonable levels.

*Table 4.8 - The comparative values of standard deviation for the total relative time, using analytical and the simulated methods. For the most part, the analytical method, utilizing a combination of statistical calculations and calculus based error methods, is very similar to the Monte Carlo simulated values. Although the percent difference between them is large, it appears that the values of standard deviation are within expected numerical error in the Monte Carlo simulation for all of the proposed methods. Even the values found for the current procedure, although larger in both single and two pass rooms when using Monte Carlo, appear to follow the analytical methods closely.*

| **Bolting Method** | **Two Pass Room** | | | **Single Pass Room** | | |
|---|---|---|---|---|---|---|
| | Mean | Analytical $\sigma$ | Simulated $\sigma$ | Mean | Analytical $\sigma$ | Simulated $\sigma$ |
| **Bolt on Fly – Full** | 1.15 | 0.04 | 0.02 | 1.14 | 0.04 | 0.02 |
| **Bolt on Fly – Right** | 1.64 | 0.04 | 0.04 | 1.14 | 0.04 | 0.02 |
| **Bolt on Pause – Single** | 2.20 | 0.04 | 0.06 | 1.71 | 0.04 | 0.04 |
| **Bolt on Pause - Multiple** | 1.92 | 0.04 | 0.05 | 1.43 | 0.04 | 0.03 |
| **Current Procedure** | 2.69 | 0.11 | 0.17 | 3.97 | 0.24 | 0.33 |

The final distributions generally fitted the normal distribution very well. The only exception to this rule is that of the brattice time-saved distribution. This is expected because the underlying value used to calculate the distribution is the speed of handling as opposed to the time taken to install a set length of brattice. This is an important distinction because the distribution is therefore related to the reciprocal of the random variable used to calculate it. Since the input values are assumed to be normal, anything related to the reciprocal of the input values cannot be normal. The brattice handling rates are based on limited data however and therefore have a large standard error, as discussed previously. For this reason, the distribution could vary more widely than shown. As a component of the total time saved, brattice time saved will also skew the normality of the total time saved distribution. It can be seen however, in Figure 4.15, that the brattice is too small a component to make a noticeable difference and although the total time distribution is changed, a normal distribution is still a very good approximation by inspection. Additional histograms for other bolting methods can be found in Appendix B.

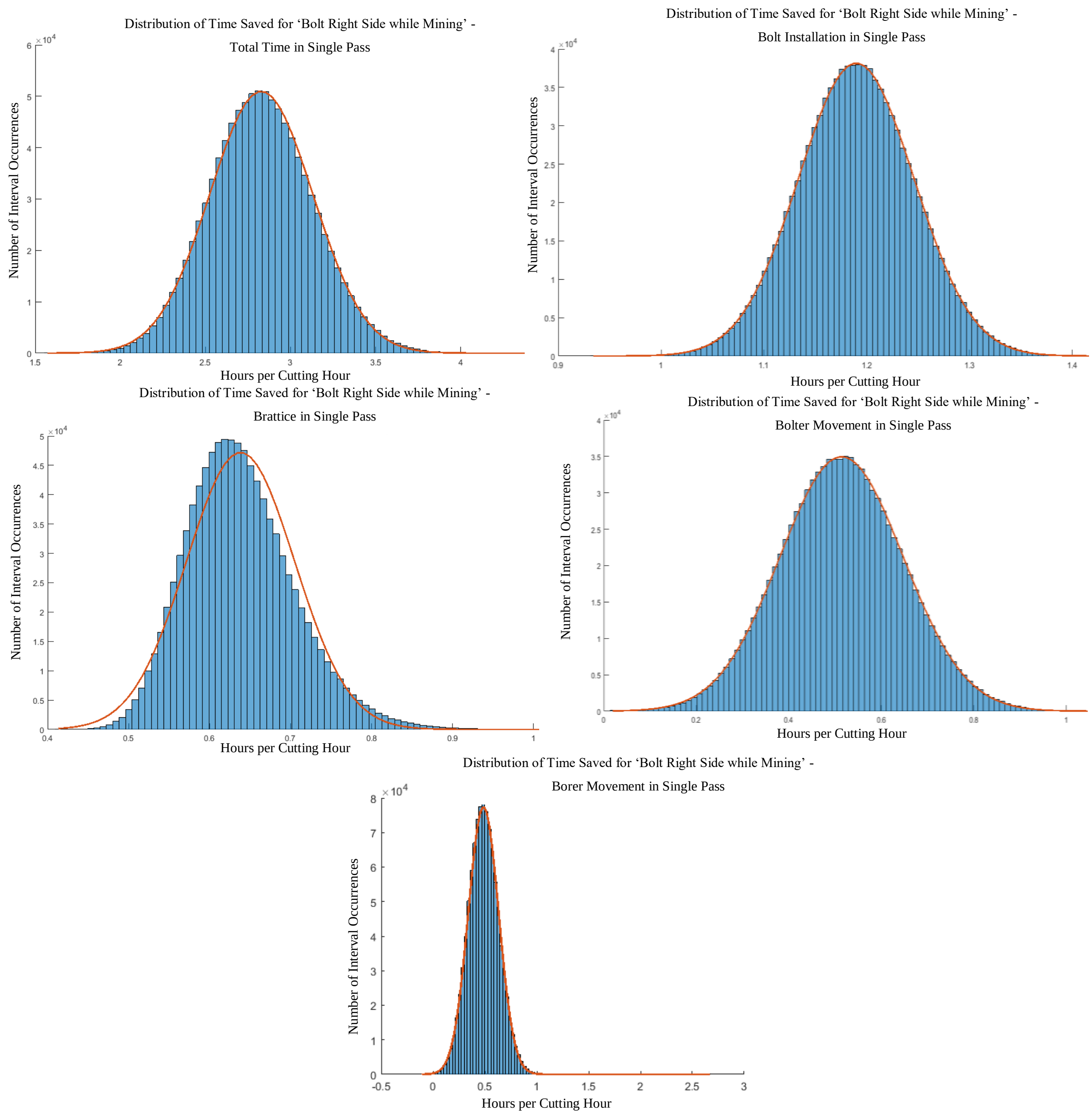



*Figure 4.15 - The histogram set for 'Bolting the Right Side while Mining' in a single pass room. The best fitting normal curve is overlaid on the histograms. Histograms developed using 1 000 000 trials. Generally, only the brattice distribution, shown middle left, differed from the normal distribution. This deviation was not enough to skew the overall time saved distribution, shown top left. All other bolting procedures closely followed this pattern while shifted slightly left or right depending on how much total time was saved by the altered procedure.*

# Chapter 5 Tunnel Stability Modelling

In addition to time improvements available, it is important to understand the ramifications of altering the current bolting procedure on the stability of the mine tunnels. Installing bolts in areas of reduced ground stability serves two purposes: prevention of blocks sliding out of the mine back, and stabilization of the mine back. Current practices involve cutting the full width of a room to approximately eight meters, stopping, and installing bolts up to the mine face. By changing to a method that allows the full length of the first pass to be cut while bolting only the right side of the pass, the location of the bolts in relation to the excavated tunnel is changed and therefore the stability is affected. Some of the proposed bolt installation procedures utilize a system where the right side of the machine is the only area ever bolted, something that is not currently done. The proposed methods also result in installing bolts during the mining of the first pass unlike the current system. One method that can be used to check the difference in stability is FEA. FEA allows the different procedures to be tested without the need to put personnel in unverified tunnels to measure displacements and stresses. It is important that there are no adverse effects on the stability of the tunnel because of the proposed bolt sequence change.

The first purpose of installing rock bolts in the mine back is to reduce the chance of pieces of rock falling into the opening. Cracks in the rock, which might be present due to clay seams or mining activities, can lead to large pieces of rock coming loose from the mine back. Failure of blocks tends to happen along layer interfaces in sedimentary rock mines. In addition, thinned sedimentary layers reduce the strength of salt beam that are more prone to buckling and, in turn, result in more extensive cracking. Bolts are installed through the blocks and into rock further from the opening, binding all the blocks in between together [11]. In areas with mining personnel, it is especially important to keep all blocks from falling into the mined entry.

The second purpose served by rock bolts is to enhance the large-scale stability of the mined opening. This is typical of sedimentary mines where multiple layers of rock, used as structural beams, are separated and allowed to slip on layer interfaces. Bolts installed through multiple layers of rock serve to bind the layers together [11]. This serves to increase the effective beam thickness and therefore increase the flexural strength, resistance to buckling, and stiffness of the salt rock beams around the opening. In areas of reduced rock layer thickness, adding bolts can increase the flexural strength and flexural stiffness and therefore, reduce the displacement of the mine back. Increased flexural strength can better carry loads to the abutment pillars of the room and therefore reduce the curvature of the back. The displacement and curvature of the back under loading can then be used as a benchmark for the stability of the mined entry.

For these two reasons, rock bolts are currently installed in the back in regions of reduced salt beam thickness and anomalous conditions in Saskatchewan potash mines. The current method used in PotashCorp mines is to tunnel eight meters across the entire width of the entry and stop to bolt. A faster proposed method would change the location of the bolts in relation to the tunnel as well as the number of bolts installed. To determine the stability of the proposed bolting patterns, FEA can be used to estimate the displacement and curvature of the mine back. Both values can be used as metrics for the stability of the entries, and are used to compare the current bolting scenarios to that of the proposed bolting patterns in this study.

## 5.1 Effects of Changed Bolt Pattern

Changing the bolting pattern to a faster procedure requires installing the bolts in a different sequence in relation to the mined entry. The two main effects this will have on stability are the change in unbolted span of the room and the change in distance between bolted ground and the tunnel face. The visual definitions of these terms can be found in Figure 5.1. The unbolted span refers to the distance between bolted ground, along the right side of the tunnel, and the opposite abutment. In the current bolting method, the maximum unbolted span is the full width of the two-pass tunnel. Proposed bolting methods would cause this maximum distance to be reduced by up to a third of the width of the opening. The current bolting pattern has a maximum distance to the mine face of approximately eight meters. Installing bolts from the boring machine requires that this distance be extended to approximately nine meters to have space for the bolt installation equipment behind the operator’s canopy. The effect of reducing the unbolted span counters the effect of increasing the unbolted distance to the mine face; the net of the two effects equates to the total change in stability for the mined tunnel.

Restrictions due to the geometry of the boring machine, limit the locations where bolts can be installed. The mechanical bolts used in active mining areas require, at a minimum, available

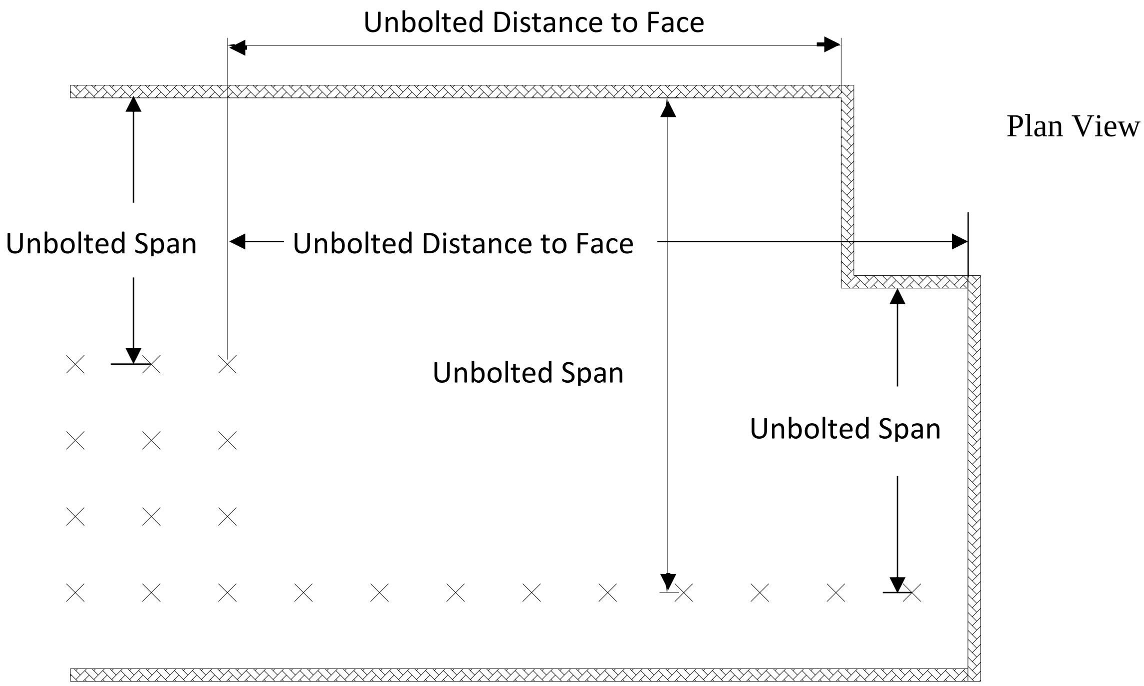



*Figure 5.1 - The unbolted span and distance to face. Bolts are installed up to the back of the borer canopy. The unbolted span of the room varies depending on the location that the measurement is taken as well as the bolt pattern spacing. Both parameters depend on the exact bolt spacing, machines used, and procedure. These are discussed in detail later in the section.*

headroom to be the length of the solid bolt shaft. This clearance does not exist at any point above the boring machines used. There also is not enough room around the sides of the boring machine to install the bolts in useful locations. Installing equipment around the sides of the machine can also impede the ability of the boring machine to maneuver in the tunnels. Therefore, the closest that bolts can be installed to the mine face is at the back of the operators canopy 9.1 meters from the mine face. There are many geometric constraints that restrict the locations where bolts can be installed from the boring machine that are discussed in more detail in Chapter Six of this thesis. The constraints include: the overlap of the boring machine in second pass, the location of the conveyor, and the location of personnel in the area, and that partial bolting systems are only able to bolt up to the right edge of the conveyor, 3.8 meters from the left pillar. These dimensions are shown in Figure 5.2.

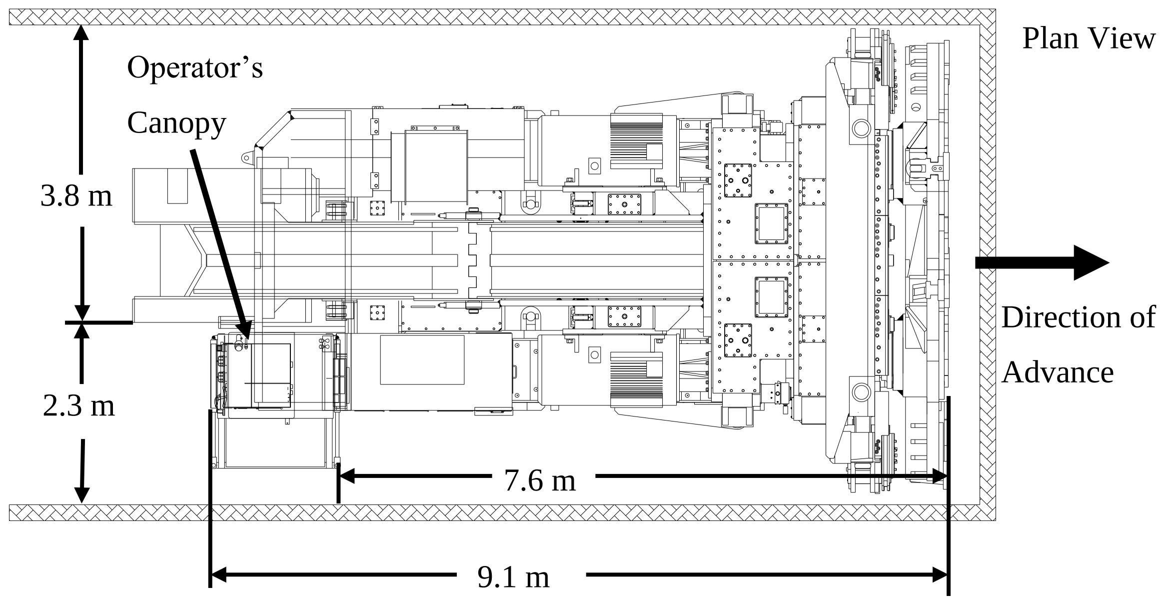
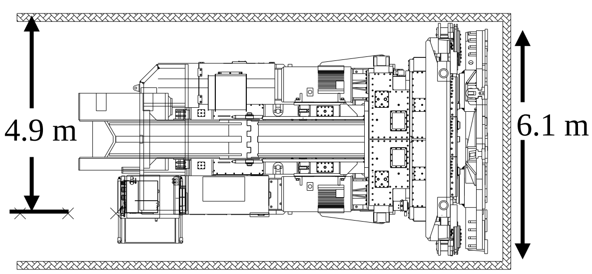



*Figure 5.2 - Geometric constraints of the boring machine for bolt installation. Installation of the mechanical bolts used by the potash mines in Saskatchewan requires more distance from the back than is available above any part of the boring machine. There is additionally not enough space beside the boring machine to install bolt installation equipment because the sides of the machine can get much too close to the sides of the tunnel at various points while mining. Partial bolting systems, and all systems in the first pass of a two-pass room, bolts installed above or to the left of the conveyor system are in the way of the boring machine when cutting the second pass. This results in bolts being cut and damage to the boring machine and therefore cannot be done. Bolts can only be installed behind the boring machine operator compartment for these reasons, 9.1 meters from the mine face and 3.8 meters from the left pillar.*

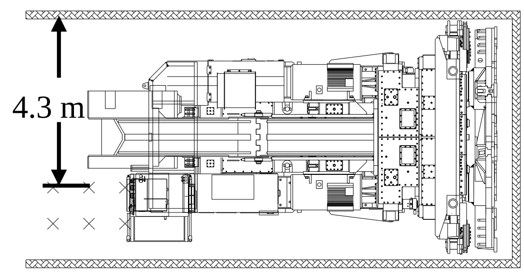
To install bolts from the boring machine, the bolts must be installed at a greater distance from the mine face than in the current bolting scenario. The current bolting scenario requires that the boring machine not advance such that the operator's canopy is under unbolted ground; this occurs at 7.6 meters of advancement. However, forward of the operator's canopy there is not enough space around the sides of the machine to install bolts in useful locations. Further, the space above the boring machine is not sufficient for installing the length of mechanical bolts used due to the stiff bolt shaft. Therefore, the closest that the bolts can be installed is the back of the operator's canopy, approximately 9.1 meters from the mine face, as compared to the front of the operators canopy as currently done, approximately 7.6 meters from the mine face.

The current bolting procedure used in anomalous ground conditions is very different from the current normal ground procedure. Instead of cutting the entire length of the tunnel, both tunnel passes are advanced 7.6 meters and the full width is immediately bolted before advancing the machine. The maximum unbolted span of the room is therefore the entire width of the room, up to approximately 9.1 meters in a two-pass room. The unbolted distance to the mine face is limited to a maximum distance of 7.6 meters. The unbolted distance can however be as short as one meter immediately after installing the rock bolts. This procedure is known to provide acceptable support levels in anomalous ground conditions.

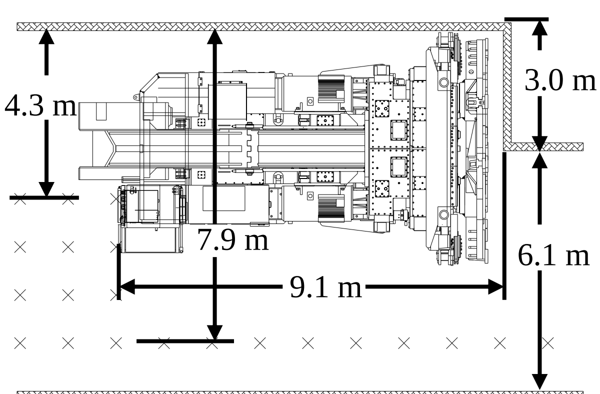
The proposed bolting procedure is based on the current normal geological ground condition procedure, which does not require bolts. The current anomalous ground cut and bolt procedure results in bolts being installed in different locations relative to the excavation of the tunnel than will be achieved with the proposed method. In the proposed procedure, the rock bolts are continually installed approximately nine meters from the mine face and up to 3.8 meters away



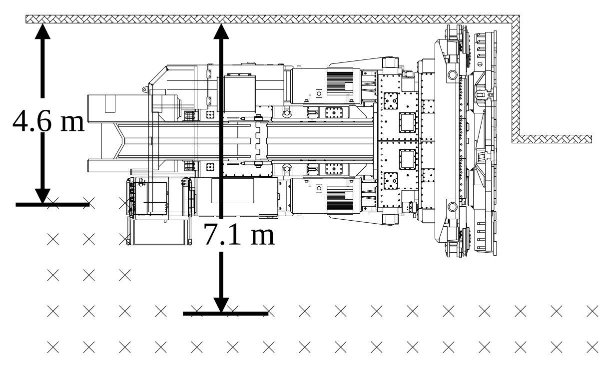

Partial bolting 1st pass
1.2-meter bolt spacing




Partial bolting 1st pass
0.91-meter bolt spacing




Partial bolting 2nd pass
1.2-meter bolt spacing




Partial bolting 2nd pass
0.91-meter bolt spacing

*Figure 5.3 - The proposed locations of installed bolts in a partially bolted ground support plan. The proposed method of bolting the entire machine width is similar except that in second pass bolts would be installed up to the wall pictured on top of the borer. Behind the borer, bolts could potentially be installed 20 cm from the rear of the boring machine. It can be seen that the bolt pattern with a 0.91 meter spacing has better coverage in both first and second pass than the 1.2-meter spaced pattern. The current bolting pattern is very different in that both passes are advanced 8 meters before installing bolts across the entire width of the entry.*

from the left wall of the tunnel. The procedure is shown in Figure 5.3. This procedure results in an unsupported span that is a stepped value. The first 9.1 meters from the mine face the unsupported span is the full width of the mined entry. In a room of single pass width, further than nine meters from the mine face the unbolted span is reduced to approximately four meters from six meters, depending on the bolt spacing used and the corresponding number of bolts that can be installed, as depicted in Figure 5.3. In a wider two-pass room, the unbolted span can take a value of approximately four or seven meters depending on both how close to the mine face it is measured, and the exact bolt spacing. Reductions in the unbolted span in some areas can increase the stability of the mined entry when compared to the same room without any bolts installed in the back.

The differences between the current cut and bolt procedure and the proposed bolt procedures in overall stability will come from the unbolted span and the unbolted distance to face. The proposed procedure results in a greater unbolted distance to the mine face than the current bolting procedure; greater unbolted distance to the mine face is expected to reduce the stability of the mined entry compared to the current procedure. The effect of the increased unbolted distance to the mine face is however expected to be offset, to some degree, by the reduction in unbolted span created by the proposed pattern. By installing some bolts while the boring machine is in first pass, the maximum span in a full two-pass room can be reduced by approximately two meters. The proposed method could also include installing bolts up to the mine face in first pass before cutting of second pass commences, limiting the unbolted span to a maximum of seven meters for the entire length of the entry. The net effect on the stability will therefore be less than the effect of either individual distance change of unbolted span or distance to face.

## 5.2 Material Properties

The material properties used for the simulations were found in previously published studies and general rules of geological materials. Two materials were used in simulating the effects of bolts on the back stability: potash, and disseminated clay. Potash is a commonly mined mineral and therefore has been extensively tested for strength properties. Potash is known to be a elasto-viscoplastic material indicating, in addition to normal elastic and plastic material properties, potash exhibits time dependent behaviours as well [17]. Over the time ranges of interest, however, the creep strain makes a very minimal difference in the total room closure, on the order of 3 mm per day [37]. Mining a short length of an entry may occur over one or two hours, and therefore allow approximately 0.25 mm of creep closure in the entry. A similar material model was used for clay. A Mohr-Coulomb plastic material model was therefore used for the purposes of finding the immediate deformation of the tunnel and creep was not considered.

### 5.2.1 Material Properties of Potash

The unconfined compressive strength of the potash material has been found to be 28.1 MPa with a standard deviation of the mean of 1.6 MPa from Lanigan potash samples [7]. The Young's modulus can also be obtained from the same paper, 5.3 GPa and standard deviation of the mean of 1.10 GPa. Additionally, as per PotashCorp internal data, the friction angle of the ore material in standard ground conditions is approximately 35$^{o}$. The final displacements that result can then be compared to the actual displacements recorded in PotashCorp mine sites of between 30 and 40 mm closure in first pass and between 40 and 50 mm displacement in second pass [37].

The material model used in the commercially available Rocscience® programs RS$^3$ and RS$^2$ is based on the isotropic Mohr-Coulomb plastic material criterion using tensile strength, friction angle, cohesion, and Young's modulus. Although potash is known to be anisotropic, an isotropic material model was used for this study. Due to a lack of data, some of these values are based on standard assumptions [38]. From the unconfined compressive strength value, the standard tensile strength and cohesion values can be calculated via the following relations:

$$\sigma_t = \frac{1}{10}\sigma_c \tag{5.3}$$

$$\tau_c = \frac{1}{4}\sigma_c \tag{5.4}$$

where $\sigma_c$ is the unconfined compressive strength

$\sigma_t$ is the tensile strength, and

$\tau_c$ is the cohesion strength.

The after yield, or plastic, parameters for friction angle are assumed to be unchanged from the original values. The Young's modulus is reduced to zero upon failure. The after-yield values for strength for both the tensile and cohesion strengths however, will be less than the original, before failure, values and were estimated to be reduced by 50%. This ratio of before to after failure values were adjusted to create a model that fits the closure data presented by PotashCorp [37]. The final values used are shown in Table 5.1.

*Table 5.1 - The final properties of potash used for the FEA simulation of the potash tunnel. The values for the friction angle, $\boldsymbol{\theta_f}$, the Young's modulus, $\boldsymbol{E}$, the tensile strength, $\boldsymbol{\sigma_t}$, and the cohesion strength, $\boldsymbol{\tau_c}$, are defined previously. The plastic material values, denoted by 'After Yield Value', were found by tuning the ratio between original and residual values from the original 50% to 86% to match experimental results.*

| Potash Parameters | Before Yield Value | After Yield Value |
|---|---|---|
| $\sigma_t$ | 2.8 MPa | 2.4 MPa |
| $\theta_f$ | 35$^o$ | 35$^o$ |
| $\tau_c$ | 7 MPa | 6 MPa |
| $E$ | 5.3 GPa | 0 GPa |
| $\sigma_{UCS}$ | 28 MPa | 24 MPa |

### 5.2.2 Clay Zone Material Properties

The material properties of the clay seam are not easily determined because the thickness of the seams in the potash ore zone are not thick enough to properly test mechanically. The disseminated clay zone in the area of interest is actually several seams of clay interspersed with halite, where none of the layers are thick enough to properly test by themselves. For this reason, effective material properties for this zone were determined by tuning the material properties to match the expected amounts of tunnel closure. Equations 5.3 and 5.4 can once again be used leaving only the compressive strength and the Young's modulus to be determined. Young's modulus here is assumed to be the equal to that of the potash ore. Based on information from PotashCorp, the displacement was graphed for a wide range of values for UCS and the value closest to the expected value (while still resulting in plausible displacement) of 8 MPa was used [37].

The modelled room displacement is affected by the strength of the disseminated clay zone material via what appears to be an inverse power relation. The room closure amounts shown with very low clay zone material strength become very large compared to the actual values expected, as shown in Figure 5.4. A final UCS value of 20 MPa was chosen because this was the lowest value where the displacement trend line still seemed to follow the known room displacement values. The lowest value was used as an attempt to represent the known weakening effect of the disseminated clay while still modelling realistic deformation.

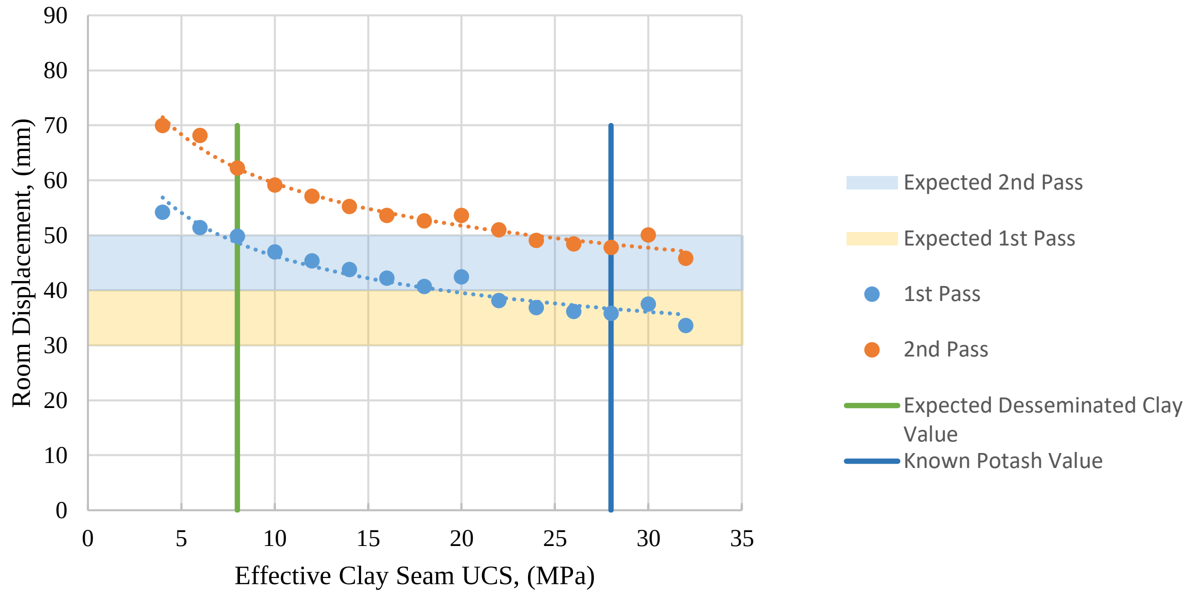



*Figure 5.4 - A comparison of the room displacement with varying effective clay seam strength values. The vertical green and blue lines represent the expected or known values of clay and potash strengths respectively. The upper trendline represents the displacement of a two-pass width room while the lower represents a single pass width. The points at 6, 20, and 30 MPa appear to have been affected by numerical errors. Further, the shaded areas represent the expected values of room displacement for a 1st pass and 2nd pass room respectively. A final UCS value of 20 MPa was used for modelling purposes because it represents the strength value closest to the expected value of clay strength while still following the expected values of room displacement.*

*Table 5.2 - The final back analyzed clay zone material properties used for the FEA simulation of the potash tunnel. The values for the friction angle, $\boldsymbol{\theta_f}$, the Young's modulus, $\boldsymbol{E}$, the tensile strength, $\boldsymbol{\sigma_t}$, and the cohesion strength, $\boldsymbol{\tau_c}$, are defined previously. The plastic material values, denoted by a 'Residual Values', found by assuming a 50% decrease from the original before failure values.*

| Clay Parameters | Before Failure Value | After Failure Value |
|---|---|---|
| $\sigma_t$ | 2.0 MPa | 1.0 MPa |
| $\theta_f$ | 20$^o$ | 20$^o$ |
| $\tau_c$ | 5.0 MPa | 2.5 MPa |
| $E$ | 5.3 GPa | 0 GPa |
| $\sigma_{UCS}$ | 20 MPa | 10 MPa |

## 5.3 Models Used

The models used for simulating the displacement of the potash tunnels allow for sequenced material removal in three dimensions. This allows for simulation of the tunnel behaviour as the tunnel is mined without calculating any time dependent effects. One of the main differences between the currently used bolting procedure and the proposed procedure centers around the sequence that the tunnel is cut making this of utmost importance. Altering the mining sequence, and when the bolts are installed within that sequence, allows the simulation to show a relative change in stability that the proposed method results in. The two procedures also differ in terms of dimensions, specifically the unbolted distance to the mine face, which will affect the actual change in displacement of the tunnel. Sequenced material removal allows both the cutting sequence and unbolted distance to be accurately simulated.

The bolts used in the model for support of the potash across the modelled clay seam are 1.83-meter point-anchored bolts. Such a bolt accurately represents the mechanical bolts used underground. There is no friction or other force modelled along the length of the bolt; equal and opposite point forces are applied at the two ends of the bolt. This allows the bolt to stretch over the entire length of the shaft and support rock across a clay seam that is anywhere along the length of the shaft. The bolt used crosses the clay seam and therefore provides support for all clay seam heights tested.

Two different cutting sequences were used to determine the relative displacements of the various cut and bolt procedures and are shown in Figure 5.5. The proposed bolting patterns use the current normal ground cutting procedure with bolts being installed up to 2.1 meters from the right adjacent pillar as cutting progresses in the first pass. For the purposes of the model, the potash ore is cut out in 9.1-meter increments. The bolts are installed in the previous block concurrently to the current ore block being cut. The bolts are assumed to apply a constant 50 kN force during this time. The current anomalous ground bolting procedure uses a slightly altered cutting pattern. The

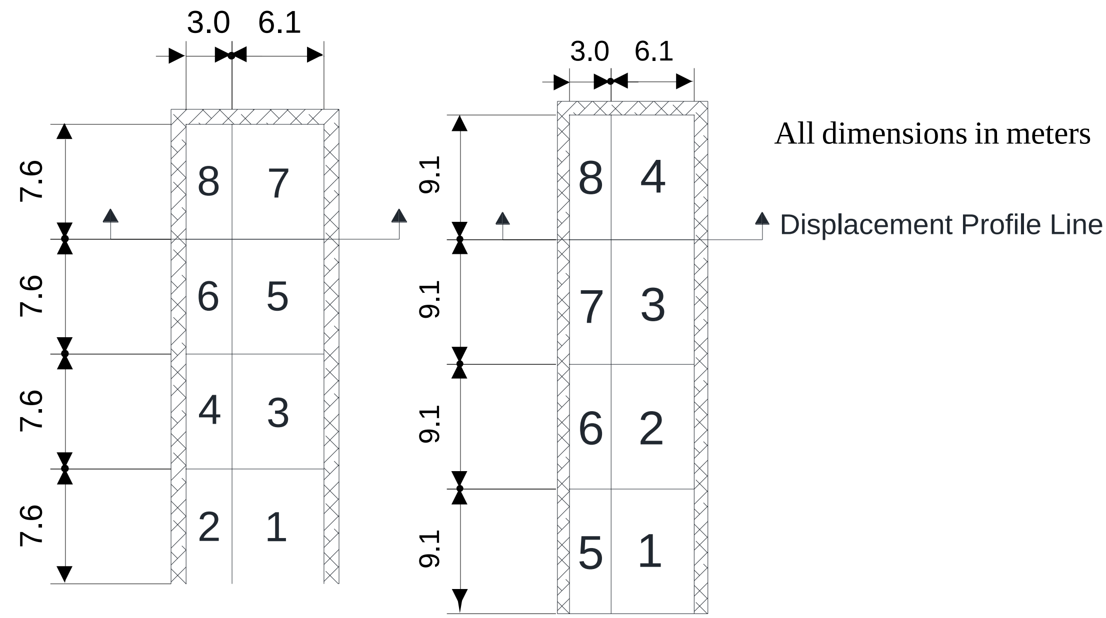



*Figure 5.5 - The plan view of the ordered cutting sequences analyzed for the current anomalous ground bolting procedure, on the left, and the proposed and unbolted procedures, on the right. All dimensions are in meters. The numbers represent the order used to remove the different potash sections. On the left, the current bolting pattern removes the total width of the entry before proceeding down the tunnel. This is unlike the proposed method, on the right, where the entire length of the tunnel is cut in the first pass before proceeding to widen the tunnel to its full finished width. The increased length of each section in the proposed pattern reflects the difference in the minimum distance between the installed bolts and the mine face. The line where the displacement profile is taken is also marked above. In both cases, it is one cutting section from the end of the finished tunnel, 7.6 meters in the current procedure*

tunnel in this case is only advanced 7.6 meters at a time. Once both the first and second passes are cut, bolts are installed before allowing any further cutting. All the cutting methods analyzed use one of these two cutting sequences.

Due to the differences in cutting patterns, the dimensions along the length of the tunnel vary. In the case of the unbolted tunnel and the proposed bolting patterns, the potash ore is cut out in 9.1-meter segments while the current anomalous ground pattern removes only 7.6-meter lengths. The dimensions of both tunnels are shown in Figure 5.5. Both patterns cut four distances into the tunnel, resulting in 36.4 meters in the proposed methods and 30.4 meters in the current anomalous ground bolting procedure from the beginning of the tunnel. Four cutting segments were used because it was found that the maximum displacement of the entries had converged to a relatively constant value at this distance from the beginning of the tunnel.

The frontal view of the tunnel is unchanged between all the various proposed and current bolting methods. The dimensions of the first tunnel pass are 6.1 meters wide by 3.7 meters tall. The second pass is situated along the left edge of the first with dimensions of 3.0 meters wide and

3.7 meters tall, as shown in Figure 5.6. The final resulting excavation is 9.1 meters wide by 3.7 meters tall. Surrounding the tunnel is potash material with one clay seam situated above the tunnel back. Tests were run with both a typical clay zone height of 0.76 meters above the top of the tunnel and a low clay zone height of 0.46 meters above the top of the tunnel. These dimensions are shown in Figure 5.6. The disseminated clay zone is 0.37 meters in thickness, as is considered normal in the PotashCorp mines. The material above the disseminated clay zone is modelled completely as potash ore. These dimensions were used with both the low clay seam and normal clay seam height for each of the bolting methods analyzed.

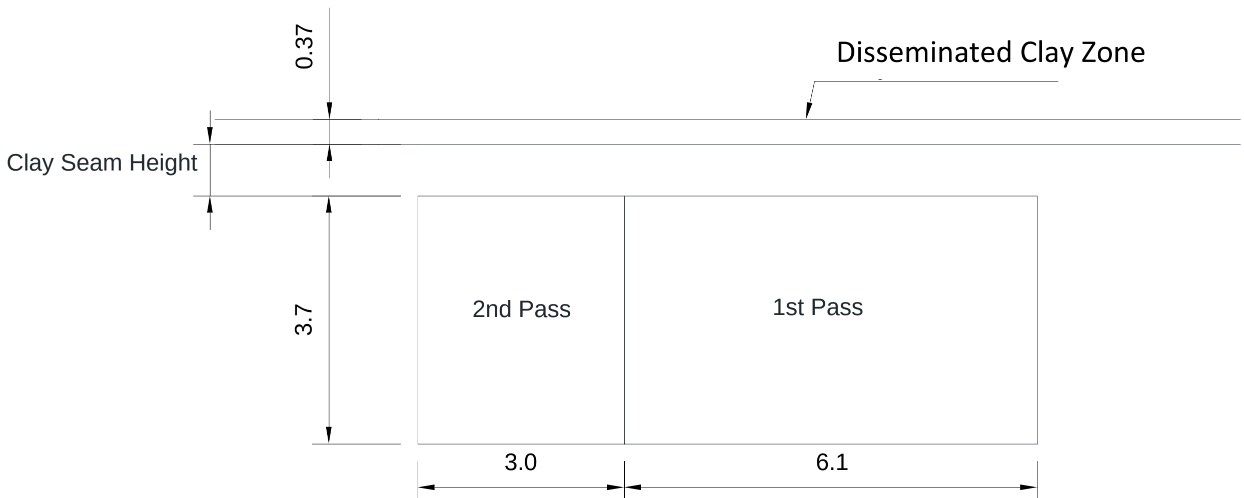



*Figure 5.6 - The front view of the model tunnel. The completed entry, the areas marked 1st pass and 2nd pass, is a total dimension of 3.7 meters high by 9.1 meters wide. The model uses a single clay seam of a uniform 0.37 meters thick comprised of a lower strength material such that the potash beams can slide past each other. The height of the clay seam varies from model to model, typically either 0.46 meters, to simulate a low hanging clay seam, or 0.76 meters, to simulate a normal clay seam height.*

Minimizing the influence of the model limitations can be difficult due to the geometry involved. Specifically, in the clay zone volume, the thin flat shape creates a possibility of elements with very high aspect ratios, an element with limited thickness but wide lateral extent, which is known to cause some numerical errors and instability. High numbers of elements are needed to limit the lateral extent of the individual elements and reduce the aspect ratios. It was found that by increasing the nodes along the exterior of the excavations such that the meshing engine resulted in approximately 54 000 elements, there were very few bad elements that would result. Additionally, these bad aspect ratio elements were concentrated around the exterior edges of the excavation so elements are not as large and accuracy is upheld. The relative lack of elements in the vertical direction within the clay seam additionally leads to the model being artificially stiff and therefore limiting the displacement. To combat this, the elements used are 10-node tetrahedrons that allow a curvature within the element, unlike a 4-node, first order, tetrahedron element type. Finally, an additional material section, beyond the end of the tunnel, that was never disturbed was used to allow the shape of the end of the tunnel to deform and properly model the end effects of the tunnel. In addition, the tunnel model used is a continuum model that does not allow for air gaps to be opened within the model materials. This last point may not be accurate in all situations although mine planning is typically done is such a way so as to avoid such gaps from forming as much a possible. Additional details are available in Appendix C. All of the first three of these measures increased the computational requirements of the model and therefore were used in moderation.

## 5.4 Comparison Method

From the results of the FEA simulations it was possible to determine the relative stability of unbolted, partially bolted, and current cut and bolt scenarios. In plastic materials, such as the potash surrounding the tunnels, the stability of the tunnel was determined based on the strain in the material from the simulation. Geological materials do not generally undergo strain hardening; they typically get weaker after material yield. This renders the final stress in the material a poor indicator of the stability because, in highly strained areas, the stress supported will be less than in un-yielded rock. This is unlike the elastic properties of many hard rocks that do not undergo large plastic deformation.

The material strain is related to several things including the displacement of the material as well as the back. The maximum displacement will increase proportionally as the strain increases in a fixed length of material and is therefore a good indicator of the overall stability of the mine room. Curvature is a better indicator of the local structural stability than the maximum displacement. As a beam curvature increases, the strain along the top and bottom edges increase; the top in compression and the bottom in tension. In the case of the potash tunnels the potash beams will be convex down into the tunnel resulting in increased tension strain in the tunnel back. Geological materials are generally very weak in tension, meaning that high curvature indicates an increased potential for failure. Both indicators were examined in determining the integrity of the finished tunnel.

## 5.5 Results

The results of the FEA were used to compare the overall and local stability of the excavated potash tunnels. A displacement profile taken near the end of the tunnel was used to show both the progression of deformation as well as the final stability across the breadth of the tunnel. Several important parameters related to stability were determined, including: the relative importance of unbolted span and unbolted distance to the face, the overall displacement, and the local curvature of the back.

The displacement profiles are determined one cutting block from the end of the tunnel as shown in Figure 5.5. It can be seen in Figures 5.7 to 5.9 that the displacement profiles tend to stay the same for several stages and then change quickly, followed by several stages that stay relatively constant. This happens when the material removed is far from where the displacement profile is taken. The last stage will be used for determining the overall displacement because it will represent the maximum displacement expected. What is currently used as the guideline for an acceptable safety level is indicated by the displacement of an unbolted room with a clay seam height of 0.61 meters as is shown in Figure 5.7.

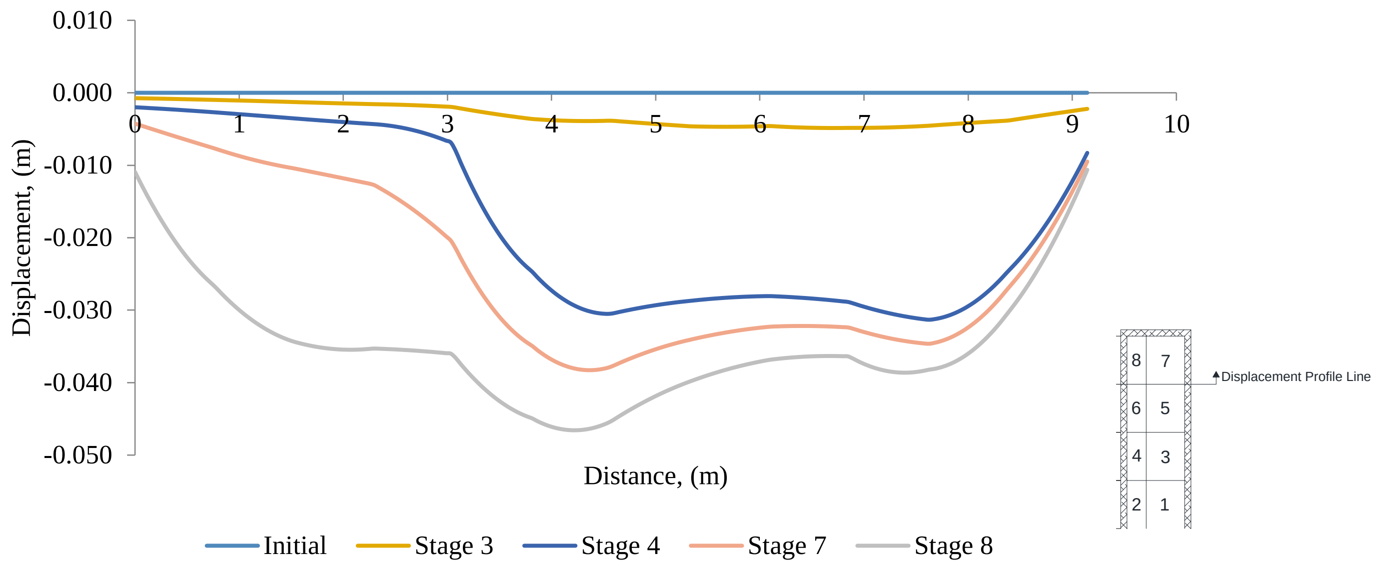



*Figure 5.7 - The tunnel back profile at 9.1 meters from the end of the mined entry for an unbolted entry and a salt beam thickness of 0.61 meters. The stages correspond to the material blocks shown earlier in Figure 5.5 (inset), with the displacement profile taken 9.1 meters from the end of the finished entry. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. The overlapping stages are removed for clarity. The location of the canopy in first pass is between the 7 and 8 meter marks, while in second pass the canopy is located between the 4 and 5 meter marks. These two locations are the areas of highest curvature. The final displacement is from the tunnel when both passes are mined out resulting in the maximum displacement. With a 0.61-meter-high clay seam the displacement is mostly smooth across the width of the mined entry. The maximum displacement at this cross section is 42.8 mm from the neutral position. This scenario is currently used as a guideline for the minimum acceptable safety level for mining practices.*

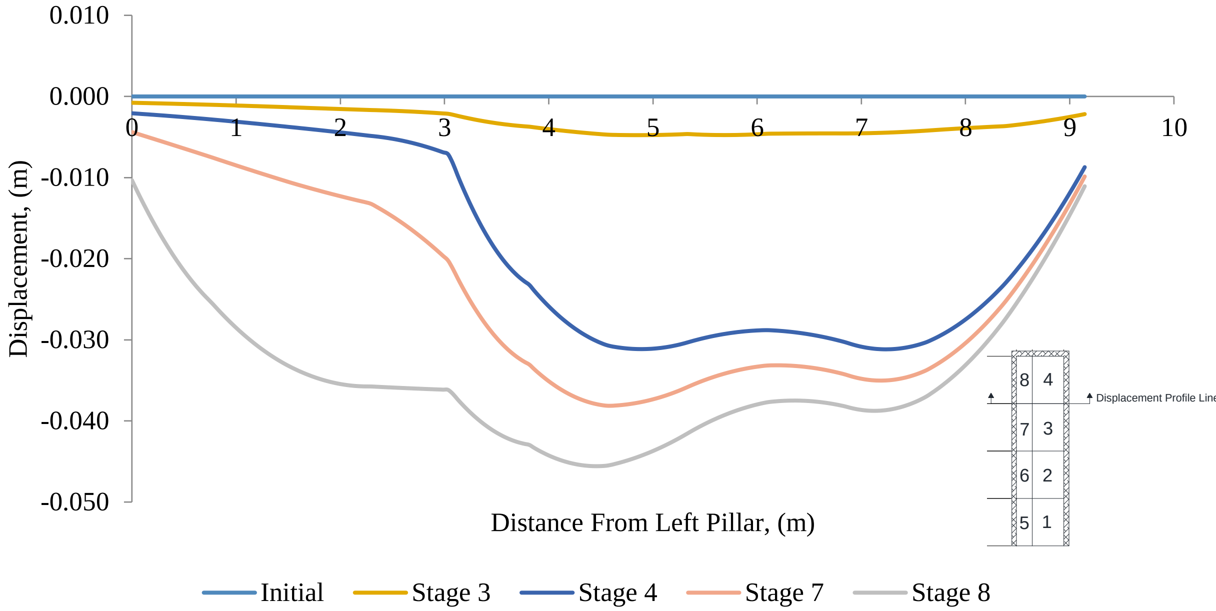



*Figure 5.8 - The tunnel back profile for unbolted ground and salt beam 0.46 meters thick. The profile is taken 9.1 meters from the end of the finished entry. The stages correspond to the material blocks shown earlier in Figure 5.5 (inset). Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. The overlapping stages are removed for clarity. The location of the canopy in first pass is between the 7 and 8 meter marks, while in second pass the canopy is located between the 4 and 5 meter marks. Two areas of high curvature are present in later stages, at 4 meters and 8 meters from the left pillar, indicating higher probability of failure at these points. In*

Variations in the unbolted span and unbolted distance to the mine face make a smaller difference than the variability of the clay seam height. The results of the various bolting scenarios are presented in Table 5.3. The unbolted distance to the mine face is shown to make the biggest difference in the total displacement of the mined opening, as evidenced by the displacement reduction achieved by the current bolting pattern compared to all proposed bolting patterns. The difference in the unbolted distance to the face between the current and proposed bolting methods is 20%, resulting in an increase from 52.3 mm displacement in the current bolting pattern to between 53.0 mm and 54.0 mm depending on the exact proposed bolting pattern when the clay seam is located 0.46 meters above the tunnel back. The maximum displacement of unbolted ground, with a similar clay seam, is similar to the partial bolting scenarios. The overall change in displacement is however very minimal, indicating a minimal change in the overall stability of the mined openings. Furthermore, the change in unbolted span of the room between the 0.91 meter spacing and 1.2 meter spacing, of 0.61 meters, makes a negligible difference in the overall stability of the tunnel. The maximum displacement therefore only minimally differentiates the current anomalous ground bolting procedure, current unbolted procedure, and the proposed bolting methods. Other methods must be used to determine the relative stability of the mined openings for mining personnel.

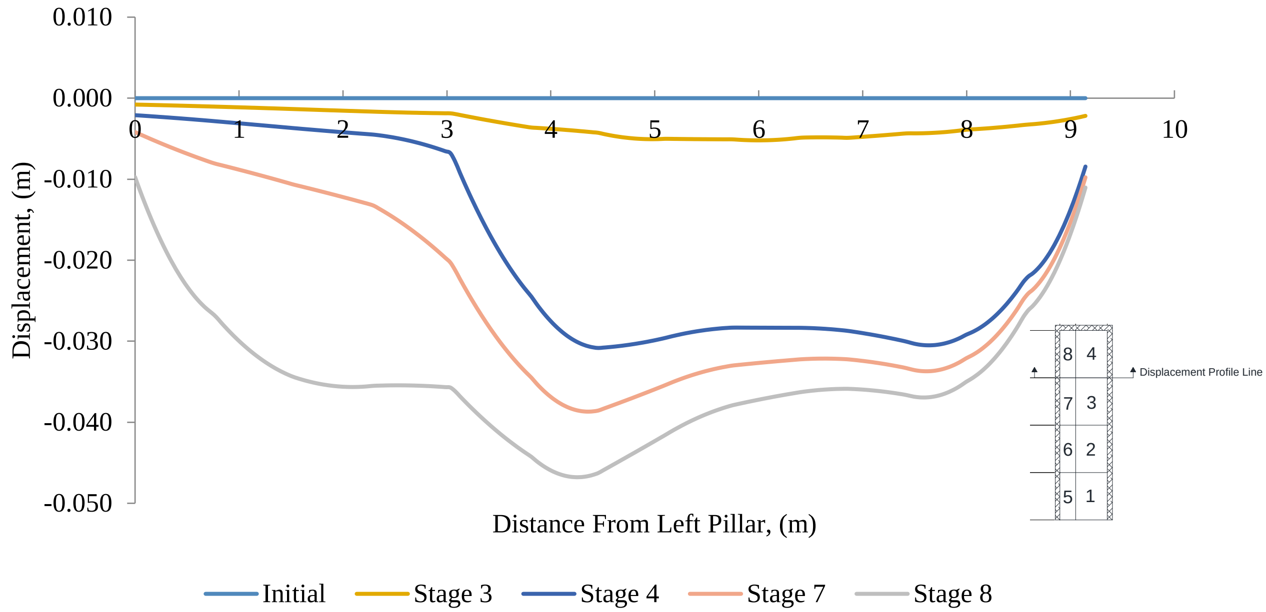



*Figure 5.9 - The tunnel back profile for partially bolted ground with bolts spaced 0.91 meters and salt beam thickness of 0.46 meters. The profile is taken 9.1 meters from the end of the finished entry. The stages correspond to the material blocks shown earlier in Figure 5.5 (inset). Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. The overlapping stages are removed for clarity. The location of the canopy in first pass is between the 7 and 8 meter marks, while in second pass the canopy is located between the 4 and 5 meter marks. The two areas of high curvature present in unbolted ground with low clay seam height, at approximately 4 and 8 meters from the left pillar, are both reduced. The bulged area at 8 meters is more noticably reduced*

On the right side of the mine tunnel, where the mining personnel are located, displacement shows more differentiation between procedures. The roof profile across the width of the tunnel can be seen in Figures 5.7 to 5.9 above for several different bolting methods and clay seam heights. Although partial bolting still does not achieve the level of stability that the current bolting pattern achieves, there is some improvement in the amount of displacement allowed along the right side when compared to an unbolted scenario on the right side of the tunnel. Partially bolting with a bolt spacing of 0.91 meters allows somewhat less back displacement to result from a clay seam height of 0.46 meters as would be expected from an unbolted tunnel with a clay seam 0.61 meters above the mine back. Similar displacements indicate similar levels of stability and therefore the indication is that partially bolted tunnel could achieve satisfactory stability at lower clay seam heights than an unbolted tunnel on the right side of the tunnel where mining personnel are located.

An additional indication of stability is the curvature of the mine back. Elevated levels of curvature can be shown to indicate increased local stress levels and therefore a higher likelihood of failure [9]. Interestingly, in Figure 5.8, the mine back displacement profile for the final cutting stage shows two areas of high convex curvature for an unbolted tunnel with a low clay seam height of 0.46 meters. In both areas, a bulge in the displacement graph is seen. These areas are located between 7 and 8 meters from the left pillar, and between 3 and 5 meters from the left pillar, the same areas that the operator’s canopy is located in first and second passes, respectively. The first such area, between 7 and 8 meters from the left pillar, is likely due to a stress concentration near the corner of the potash tunnel combined with increased displacement resulting from the first cutting pass. By comparison, this right most bulge is almost eliminated when partial bolting with a 0.91 meter spacing, as shown in Figure 5.10, and curvature is correspondingly reduced. Partial bolting results in nearly the same displacement reduction as the current bolting method in this high curvature area. The curvature in the middle of the finished room is not affected as much by partial bolting as the curvature on the right side. Comparable results are found for all other proposed bolting solutions, the right bulge is almost eliminated with the curvature reduced while the middle maximum curvature is only reduced by less; the graphs for other bolting methods can be found in Appendix D. The maximum curvature locations show that partially bolting the mine back when low clay seams are present increases the stability of the room in areas where mining personnel are required.

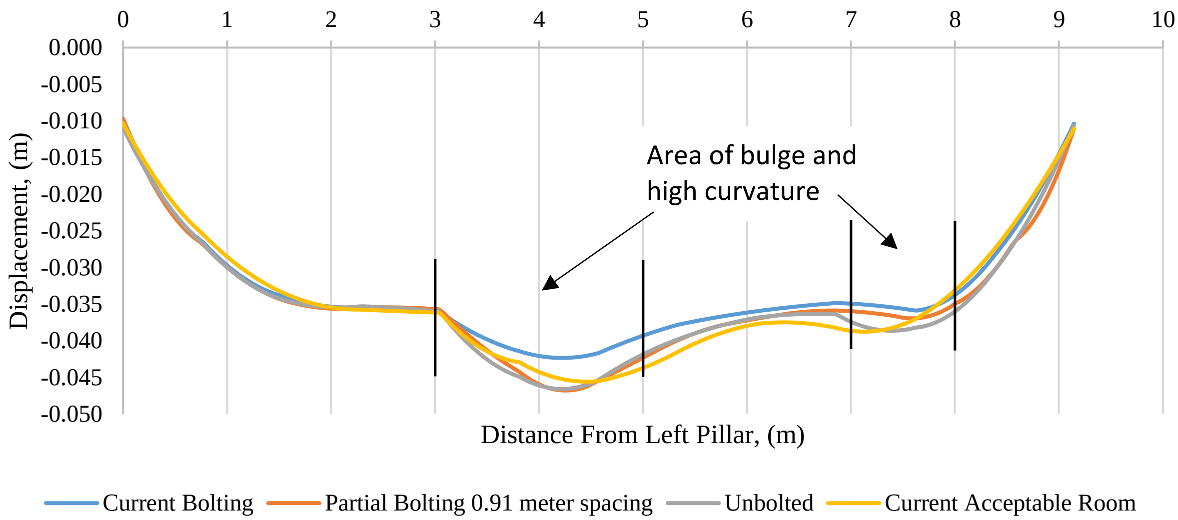



*Figure 5.10 - A comparison of the final room displacements for three of the analyzed bolting patterns in a low clay seam height of 0.46 meters and a currently acceptable room. Note that the current bolting pattern profile is taken closer to the mine face because of reduced distance being unbolted and therefore shows less displacement. The areas between 7 and 8 meters, and 4 and 5 meters, correspond to the places the operator's canopy is situated in first and second passes. Stability improvements in the area of high curvature closer to the right pillar are shown by partial bolting. All proposed bolting methods yielded very comparable*

In Figure 5.11, the numerical values of curvature of the same scenarios shown in Figure 5.10 are displayed. Due to the limited number of elements, the resolution is limited. In Figure 5.11, it is seen that even when using approximately 54 000 elements for the simulation, there are only approximately 12 to 14 elements spanning the width of the tunnel at any cross section. The nature of second order elements is that the curvature is constant along the element edge, resulting in 12 to 14 constant values for curvature across the width of the tunnel. Accuracy is further hindered by the existence of stress concentrations in the corners of the tunnel, making accuracy close to the edges of the graph questionable. Comparing to Figure 5.10, the differentiable areas of the graph are related to the two bulges in the back displacements between three and five meters, and between seven and eight meters from the left pillar. Other areas are either too close to the stress concentrations at the corners of the room or are too similar to each other to differentiate.

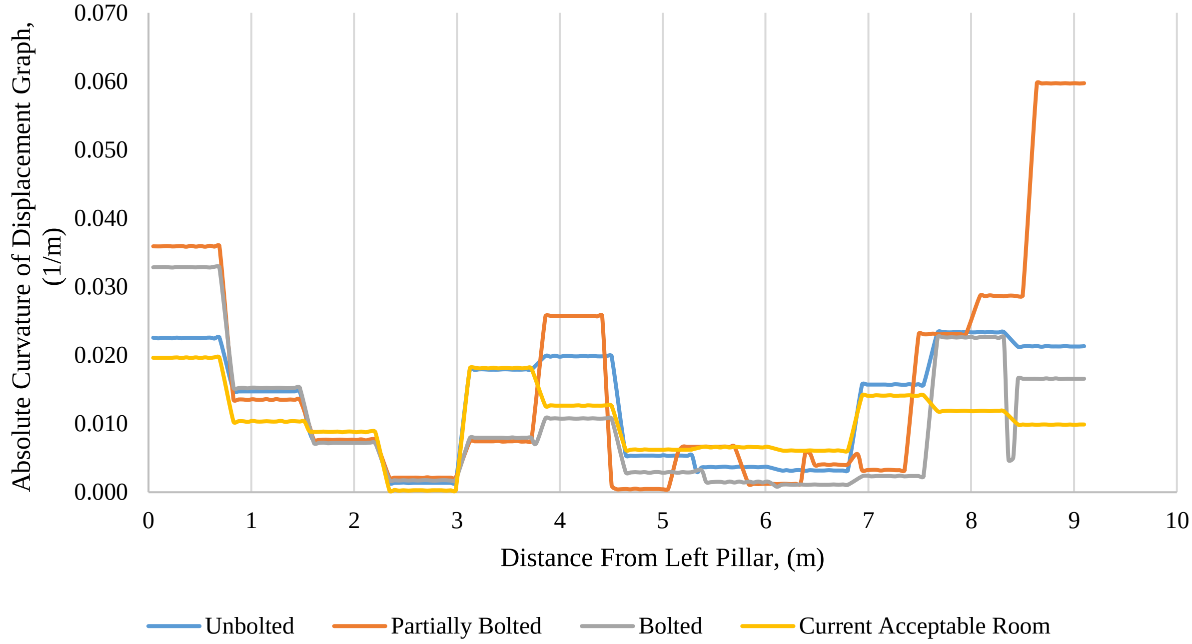



*Figure 5.11 - Plot of the numerical curvature of the mine back across the width of a room two passes wide for different bolting methods. The elements used to simulate the displacements are second order elements, resulting in a constant curvature across an element edge. As such, the curvature at the boundary between two elements is not valid because this will increase as the distance between discrete points decreases. The values at these boundaries were therefore made removed for the purposes of this graph. Per the information shown in Figure 5.10, the two areas of interest are in the range of approximately three to five meters and seven to eight meters from the left pillar. These areas are in the two displacement bulges in the graph and are far enough from the corners and associated stress concentrations to be considered accurate. The average of the two elements spanning the region of seven to eight meters in partially bolted back is approximately 0.0132/m, marginally greater than that of the currently acceptable room of 0.0130/m. Both are greater than that of the current bolted procedure of 0.0125/m and much less than that of 0.0195/m.*

Average curvature values in each of the areas of interest in all of the bolting patterns analyzed can be found in Table 5.3 using the displacement profiles generated from simulated results. The partial bolting techniques tend to be comparable to the current minimum acceptable unbolted ground scenario for the region above the operator and other underground personnel. In the region between three and five meters from the left pillar, partial bolting patterns are comparable to unbolted ground conditions. The mean curvature of the two elements spanning between seven and eight meters from the left pillar for the partial bolting pattern with 0.91-meter bolt spacing is 0.0132/m. This compares to 0.0125/m for the current bolting technique, 0.0130/m for the current acceptable ground condition, and 0.0195/m for the unbolted scenario. For this region, the curvature is very similar between the partial, and current bolting patterns as well as for the current allowable condition. All three methods allow much less curvature in this area than the unbolted simulation. In the region between three and five meters, the values of curvature are 0.0144/m, 0.0123/m, 0.0112/m, and 0.0072/m for the unbolted scenario, current acceptable ground condition, partial bolting method, and current bolting method respectively. In this region, the curvature is again similar between the current acceptable ground condition and the partial bolting method. Both methods allow much less curvature than the unbolted case and much more curvature than the current bolting method. Other partial bolting methods result in similar curvature values in these two areas and are therefore comparable to the current acceptable ground condition but less stable than the current bolting method. This verifies what can be seen in Figure 5.10.

Partially bolting a room in an underground potash mine can be shown to increase stability of the tunnel systems when compared to an unbolted tunnel. Partial bolting of a tunnel reduces the unbolted span of the room but, to bolt behind the borer, the distance between the bolts and the mine face is increased from current anomalous ground bolting procedures. The three stability indicators that were explored show mixed results for partial bolting. The maximum displacement of the room is largely unchanged between partially bolting, current bolting procedures, and not bolting at all, indicative of minimal overall stability improvements. On the right side however, the curvature and the displacement are reduced greatly in low clay seam height areas by partially bolting. One displacement bulge above the mining personnel is almost eliminated by using partial bolting patterns and the corresponding high curvature is reduced in the area. All the proposed patterns that install bolts along the right side of the tunnel before moving to cut the second pass yield almost identical results. In certain areas, partial bolting can achieve acceptable safety levels while utilizing the normal ground cutting procedure.

*Table 5.3 - The end results of the finite element simulations. Current acceptable methods include the all current bolting practices and the unbolted scenarios with clay seam height of 0.61 meters and greater. Very minimal differences exist in the total displacement values, indicating that material yield has not occurred to a substantial extent in the clay seam. With a lower clay seam strength, it is possible that differences would be observed. The proposed patterns generally result in the same displacements whether partially or fully bolted and regardless of bolt spacing. The curvature, particularly between seven and eight meters from the left pillar, is where partial bolting systems are comparable to current acceptable ground conditions. Virtually all the proposed methods result in very similar results with the exception of the partial bolting with 1.2-meter bolt spacing. The curvature values follow what would be expected from Figure D.9 but is considered an outlier. Curvature in the range of seven to eight meters can be considered an indicator of stability over the operator.*

| Scenario | Seam Height (m) | Displacement Maximum (mm) | | Curvature (3 to 5 meters)(1/m) | Curvature (7 to 8 meters)(1/m) |
|---|---|---|---|---|---|
| | | 1st pass | 2nd Pass | | |
| **Current Unbolted** | 0.46 | 39.2 | 53.6 | 0.0144 | 0.0195 |
| | 0.61 | 39.0 | 52.0 | 0.0123 | 0.0130 |
| | 0.76 | 42.5 | 53.7 | 0.0094 | 0.0079 |
| **Current Bolted** | 0.46 | - | 52.3 | 0.0072 | 0.0125 |
| | 0.76 | - | 49.6 | 0.0054 | 0.0088 |
| **Proposed 0.91 Meter Spacing Partial** | 0.46 | 38.5 | 53.9 | 0.0112 | 0.0132 |
| | 0.76 | 39.0 | 50.8 | 0.0048 | 0.0092 |
| **Proposed 1.2 Meter Spacing Partial** | 0.46 | 38.7 | 53.1 | 0.0182 | 0.0193 |
| | 0.76 | 38.8 | 50.8 | 0.0043 | 0.0087 |
| **Proposed 0.91 Meter Spacing Full** | 0.46 | 38.7 | 54.3 | 0.0124 | 0.0136 |
| | 0.76 | 38.8 | 50.7 | 0.0048 | 0.0091 |
| **Proposed 1.2 Meter Spacing Full** | 0.46 | 37.9 | 53.2 | 0.0156 | 0.0101 |
| | 0.76 | 38.9 | 50.7 | 0.0168 | 0.0090 |

## 5.6 Conclusions

The FEA yielded some interesting results pertaining to the viability of partially bolting rooms in underground mining conditions. Installing bolts in the tunnel was shown to reduce the total displacement by approximately five percent while varying the clay seam height yielded approximately a ten percent difference in the back displacement. Changing the location and installation patterns of the bolts also changed the total displacement in the tunnel as well as the amount of displacement along the right side of the tunnel where mining personnel are located. Much less relevant to the stability of the tunnel was the spacing of the installed bolts; both 0.91 and 1.2 meter spaced proposed alternatives yielded similar displacement results. Partial bolting techniques result in noteworthy local stability improvements along the right side of the tunnel when compared to the unbolted ground scenario based on curvature values. Stability metrics indicate that partial bolting may be adequate in certain scenarios based on the information gleaned.

Due to the staged nature of the excavations, it is expected that the deformation would happen progressively as material is removed. All the cutting patterns start the first pass along the right side of what will become the finished room and this is where the highest deformation is seen. The back profile of the current anomalous ground bolting procedure is seen to have the most gradual displacement profile of all methods analyzed. This was expected due to the nature of the cutting order change described in Figure 5.5. The current cut and bolt ground procedure excavates the full width of the pass a short distance, and allows it to subsequently displace uniformly. This is unlike all other methods analyzed that cut the entire length of the first pass at once, and allow displacement, before starting second pass. This results in considerable deformation before the second pass is excavated as shown by the final displacement profiles in Figures 5.7 to 5.9 .

The stability metrics of the tunnel, curvature and displacement, are comparable between the current acceptable ground condition and partial bolting methods in certain areas. The overall stability metrics are only changed minimally by the addition of bolts in any scenario from the unbolted ground condition. The curvature, however, is changed depending on the method. Specifically, the bulge in the displacement graph between seven and eight meters from the left pillar, present in the unbolted ground condition with 0.46-meter clay zone height, is greatly reduced by partial bolting techniques and results in similar to slightly reduced values of curvature in both bulged areas. Displacement in model results from low clay zone heights are also similar between partial bolting models and the current acceptable ground condition model. In the areas over the mining personnel, partial bolting techniques result in the same or increased stability metrics when compared to the current acceptable ground condition and less curvature than similar ground left unbolted.

Partial bolting scenarios show similar stability metrics to those realized by currently acceptable ground scenarios. The overall displacement of the tunnels is shown to be changed only minimally by installing bolts while the curvature in minimized by the current cut and bolt ground procedure. The proposed partial bolting methods achieve similar metrics to the currently acceptable unbolted ground at lower clay seam heights but not the stability metric levels shown by the current anomalous ground bolting procedure. This indicates that partial bolting may be acceptable in some areas that are currently bolted, but not all. Low-risk anomalous ground conditions, such as low clay seams and leach anomalies, could attain similar levels of curvature and displacement in work areas as other acceptable ground conditions by utilizing partial bolting techniques. The simulations indicate that high-risk anomalies, such as extremely low clay seams or where clay seams intersect the mine back, may still need to be fully bolted. Partial bolting techniques can be considered feasible in many currently bolted areas of underground potash mines.

# Chapter 6 Concept Design and Determined Machine Constraints

To determine feasible bolting patterns and bolting sequences, a machine capable of bolting from the boring machine is in development. Information on the stability and time savings available are only relevant if they can be implemented underground. A machine that will install bolts from a continuous boring machine currently does not exist so one must be developed. Machine development is focused on the ability to bolt only the right side of the machine conveyor belt due to the nature of the normal underground cutting pattern, available space, and the location of personnel with respect to the boring machine. Machine development has led to more relevant methods being analyzed for time and stability.

All machine development has been focused on a machine capable of bolting on the right side of the conveyor belt while mining is paused. Under this design scenario, anytime that bolts must be installed over top of the conveyor belt or further left, a specialized bolting machine would still need to be brought in. This corresponds to the 'Bolt while Paused – Single' bolting method for time savings analysis. There are several reasons to only bolt the right side of the machine: the current cutting pattern for normal ground conditions, and the location of underground personnel in relation to the borer. The current cutting pattern for the mines involves boring the entire first pass along the right side of the planned entry, before widening by cutting a partial second pass the same distance. To be able to cut the second pass, which is a half width of the machine, there cannot be bolts installed in the borer's path; bolts installed in the overlapping area between passes are cut by the boring machine, rendering the bolt useless and damaging the cutting teeth on the borer. Therefore, bolts can only be installed in first pass up to 3.0 meters from the right pillar in the room to avoid damage. Another reason to focus on bolting the right side of the room is to ensure that bolts are installed directly over the work area for mining personnel. Current work does not require any personnel on the left side of the boring machine or forward of the operator's canopy to

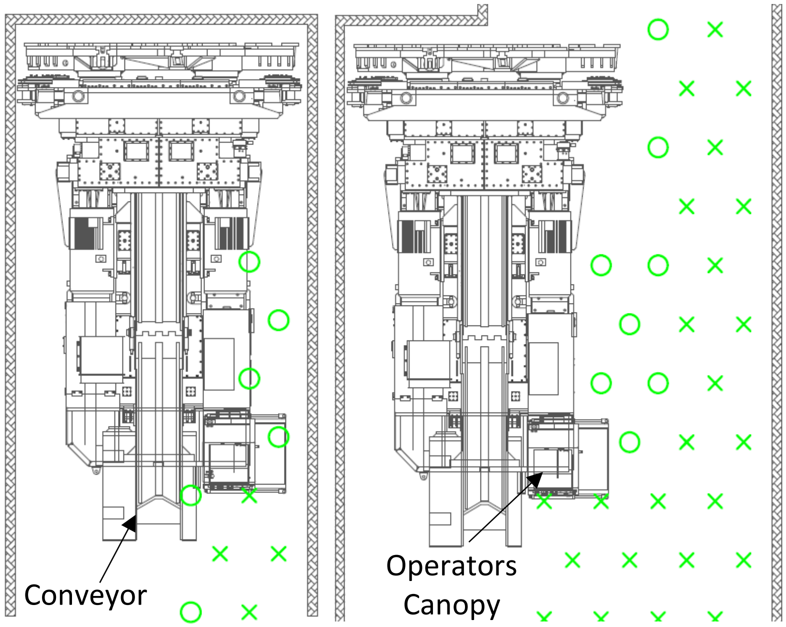


*Figure 6.1 - Bolt installation locations for a machine bolting the right side of the conveyor only. O's indicated locations where bolts may be possible while X's denote locations where bolts are expected. Locations shown are for a staggered 1.2-meter by 1.2-meter pattern. Bolts in this scenario are located only on the right side of the conveyor system.*

accomplish any task. Ensuring that ore slabs do not slide out of the back over top of work areas can be accomplished by bolting only the right side of the machine.

The bolting pattern for a borer-mounted bolter must be changed from current practice to fit a bolting system in the space behind the borer operator's canopy. Current bolting patterns, shown in Figure 4.1, are not suitable in this instance because the space required to install a second bolt is p to 2.4 meters from the right pillar. This pattern is shown in Figure 6.1 in relation to a borer for a right side only bolting system. At this distance, the bolt would have to be installed above the conveyor belt of the machine, a location where not enough head room exists to install a bolt. The space available for installing bolts behind the operator's canopy is up to approximately 1.8 meters from the right wall. One bolt in this space would leave some unbolted area along the conveyor so two bolts would be required. The bolting pattern for the concept is therefore decided to have bolts spaced 0.91 meters apart.

## 6.1 Previous Designs

As previously mentioned, there are currently no bolting machines designed to mount to a continuous boring machine. Borer-mounted bolting machines have been built but unfortunately are designed on a non-continuous boring machine; such systems will transfer to a continuous boring machine only if mining is stopped while installing the bolts. The machines currently used for bolting underground are separate, specialized bolting machines designed only to install bolts. Systems designed to drill test holes have also been installed on continuous boring machines, which although not the same, share some similar functions with a borer mounted bolting system. The methods used on these machines can be repurposed to create a borer mounted bolting machine.

Almost all the bolting machines currently available share some similarities in the methods used to install the mechanical rock bolts. The bolts are generally installed using a combination of at least two hydraulic motors, one to tighten the bolt and one to provide thrust on the bolt. Some of the machines will use four motors, two identical sets of two motors for each of the drilling and installation phases of the bolt install. Commonly, the systems are designed such that one hydraulic motor drives a chain that advances the other hydraulic motor towards the mine back. The other motor, used for tightening the bolt, is positioned using a linear slide. An issue with some current designs is the consistency of aligning the bolt to be installed with the freshly drilled hole. When these two functions are completed by separate systems and indexed, the indexing function must be accurate to approximately three millimeters to allow the bolt to slide into the hole properly.

A drilling system for drilling test holes has been attached to the continuous borers at the PotashCorp Allan site that accomplishes half of the task of installing a bolt. The system is only capable of drilling when mining is stopped so that the drilling system does not have to compensate for boring machine movement. A bolting system that is mountable to a continuous boring machine requires some similar capabilities and therefore some design characteristics can be taken from this machine. To create a bolting machine from the drilling system, an indexing system capable of changing from drilling mode to bolting mode needs to be added.

## 6.2 Requirements

The constraints for a bolting system designed only to install bolts when mining is paused are simpler than they are for a system designed to bolt while mining. Generally, there is one main constraint for a bolting system that installs bolts while the mining process is paused: that the system is compact enough to not interfere with the maneuverability of the boring machine. Further constraints exist for the machine if bolting while mining to compensate for the movement of the borer while installing the bolts. Requirements for both types of systems are otherwise the same for the types of bolts to install and bolt coverage required.

The space available to mount a bolting system to the boring machine is limited. The space chosen for the machine, directly behind the operator's canopy, is constrained while the borer is turning from one entry to another with the conveyor cars attached. The conveyor cars act as a trailer would behind a vehicle, as shown in Figure 6.2. To allow the boring machine to still turn with the conveyor cars attached, the system needs to fit within the distance left when the boring machine is perpendicular to the conveyor car. This geometry leaves approximately 0.36 meters of distance between the back edge of the operator's canopy and the edge of the conveyor car. The borer would only be turned when the mining procedure is stopped and therefore the bolting apparatus must fit in this space only when fully stowed.

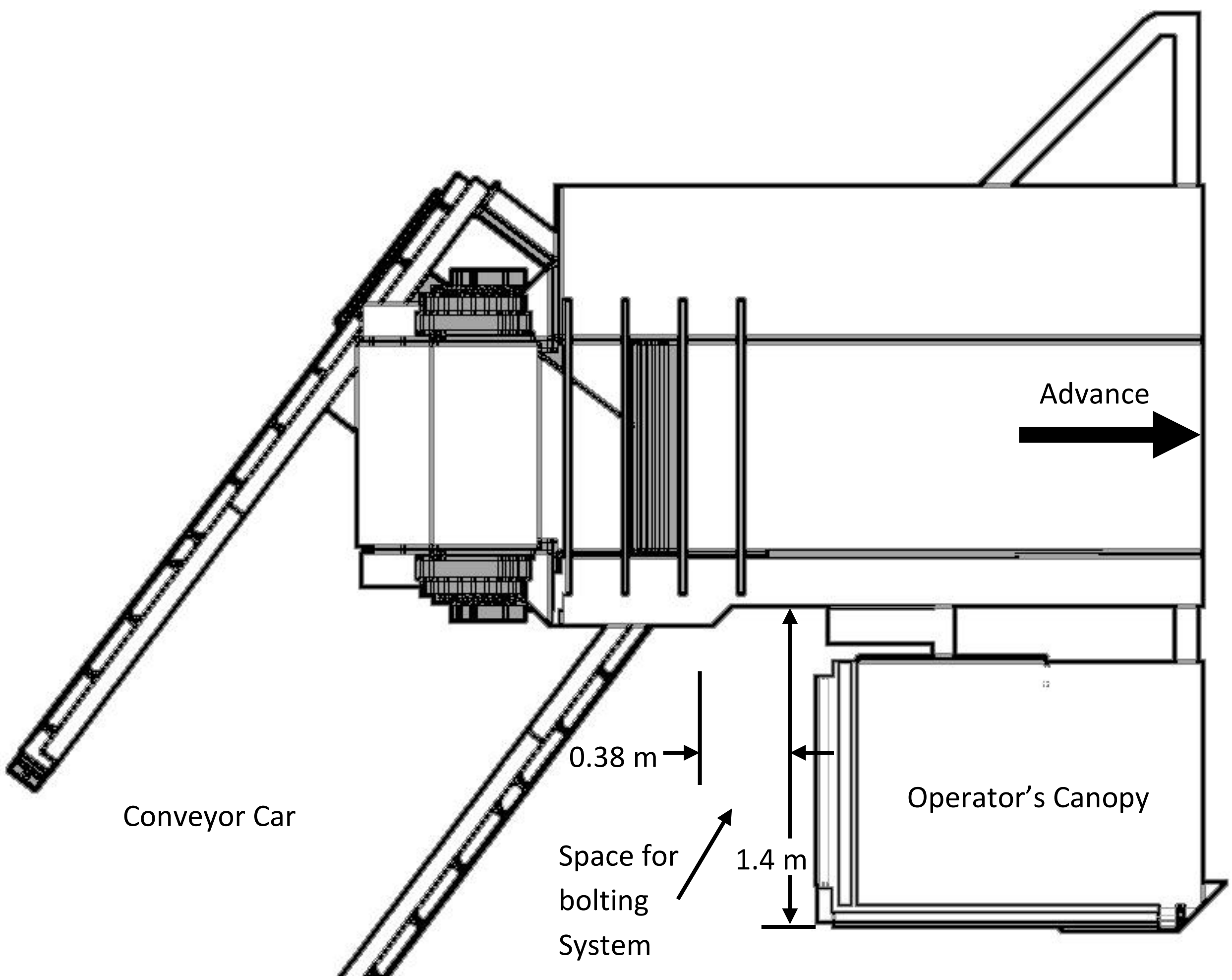



*Figure 6.2 - Top view of borer rear and conveyor car. Like a vehicle towing a trailer, as the turn angle becomes greater the conveyor gets closer to the back of the borer. The conveyor may be perpendicular to the borer in some cases.*

For the bolting machine to be able to install bolts while mining, there must be a method to compensate for the boring machines movement during the time that the bolt is being installed. Motion on all three axes need to be compensated for in both rotation and translation to allow the bolting machine to stay in a fixed position in relation to the bolt hole. The main movement to compensate for is the boring machine moving forward at approximately 0.30 meters per minute. During the drilling cycle, measured to take 71 seconds, the bolting machine must accommodate a minimum of 0.36 meters of forward movement. Other mining movements are much less noticeable. Specifically, the roll, pitch, and yaw angles all change very little over the course of 0.36 meters of mining advancement. In this amount of space, indexing between drill and bolt and installing extra features such as bolt magazines become challenges.

The borer mounted bolting machine has several other requirements regardless of the capability to install bolts while mining. The bolts installed in underground potash mines must be a point-anchored style mechanical bolt to allow for large deflection in the tunnel back without breaking. Complete bolt coverage must be attained over the areas where personnel are working. Accomplishing complete coverage with the bolting machine requires a method of extending the bolting apparatus beyond the right edge of the operator’s canopy when in second pass but completely retract behind the borer in first pass to avoid interference with the borer maneuverability.

Current bolting specifications are also required for a new machine. The bolting machines currently are capable of 440 Nm of torque as this provides adequate pre-tensioning for the bolts. Drilling specifications are not firm requirements and will likely be limited by available hydraulic system power. Current drilling systems can use 290 Nm of torque, 6.7 kN thrust force, and 860 rpm drilling speeds [37]. This allows a penetration rate of 3 m/min.

There are relatively few constraints pertaining to the creation of a continuous borer mounted bolting machine. However, the constraints that do exist cause the continuous borer mounted bolter to be a difficult engineering task. The space available makes it difficult to use some commonly used methods of indexing and multiple bolt-holding systems. Increased difficulty is created by the attempt to make the bolting system install bolts while mining is in progress. As a first step in the development of such a bolter, only a machine capable of installing bolts while mining is paused was considered.

## 6.3 General Concept Design

The machine concept developed had to be able to fulfil the constraints outlined earlier while the boring machine is paused. A single bolting apparatus was used to accomplish this task utilizing two symmetrical drilling systems, one to drill a hole and one to install and tighten the mechanical bolt. Coverage is accomplished by using a borer mount with lateral sliding rail to allow the drill system to reach any point across the width of the borer mount. The concept could be furthered to include the ability to install bolts while mining by adding degrees of freedom to compensate for the forward movement of the boring machine and one to accommodate the change in distance between the boring machine roof and tunnel back.

The bolting apparatus is designed around a standard I-beam with symmetrical equipment on the two sides. An I-beam is used because it provides increased structural stability over other methods and provides two sliding rails on each side for advancing drill motors. On each side, the system is designed with two hydraulic motors, one to advance the drill and the other to provide torque to the drill chuck, for a total of four hydraulic motors. Hydraulics were used due to the availability of hydraulic power on the boring machine. Drill feed rate is controlled by the bottom motor shown in Figure 6.3. This motor is connected to a chain drive system that can both feed or retract the drilling motor. This allows indexing between the sides of the drill via a linear sliding surface affixed to the bottom of the I-beam. This bolting apparatus therefore provides all the functionality required by the borer mounted bolting system.

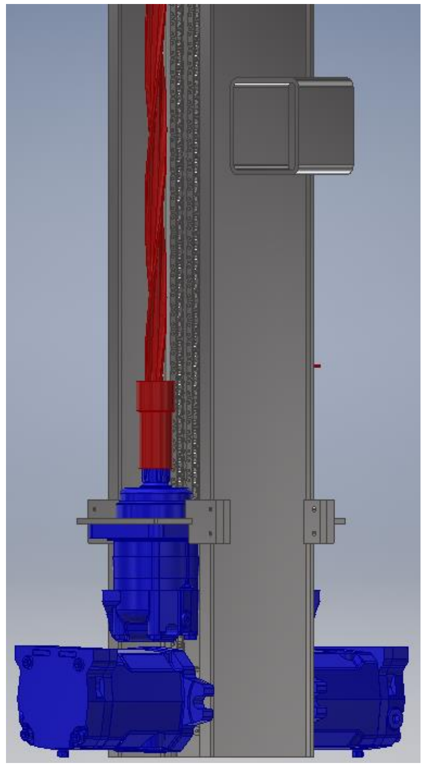


*Figure 6.3 - An image of the bolting/drilling apparatus. The blue hydraulic motor on the bottom left of the image is used to drive the chain and provide drilling thrust force. A mirror image of the setup is found on the other side, used for bolt installation and tightening.*

The borer mount for the drilling apparatus is as important as the drilling apparatus itself in allowing the borer mounted bolting system to achieve the given requirements. Mainly, the borer mount must allow lateral movement, perpendicular to the direction of advancement, so that multiple bolts can be installed across the width of the tunnel while providing a stable platform for the drilling apparatus. Lateral movement is achieved by using a lateral slide, shown in Figure 6.4, and powered by a single hydraulic cylinder. In order to complete the bolting patterns shown previously, some bolts must be installed beyond the right most edge of the operator’s canopy. An extension is required to install bolts in the area beyond the canopy while allowing the system to still fit behind the operator’s canopy in first pass. For added rigidity, the drilling apparatus is affixed to the borer mount via a platform that is able to press against the floor and back of the tunnel, effectively mounting the drilling apparatus to the tunnel instead of the boring machine, shown in Figure 6.5. The rigidity of the system will improve consistency of aligning the mechanical bolt with the previously drilled hole.

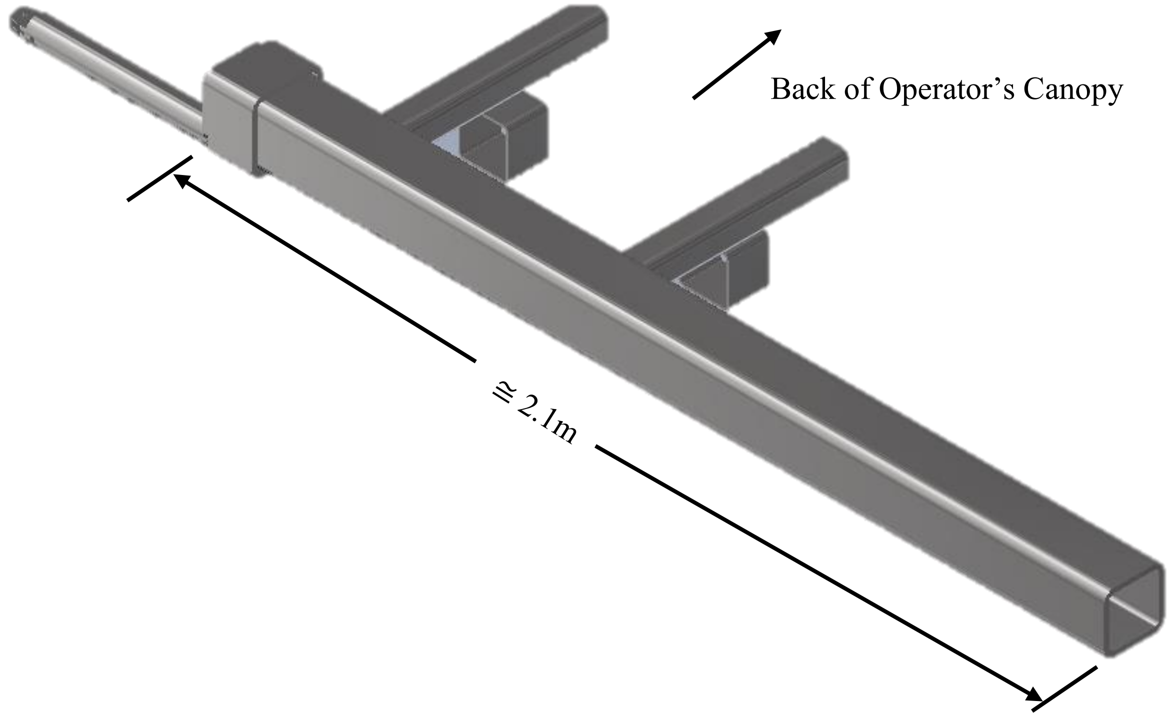



*Figure 6.4 - The designed borer mount. The system would be attached to the borer on the top right-hand corner of the image and the drilling apparatus on the slide located on the top left. The slide is moved back and forth by a single telescopic hydraulic cylinder in the top left. There is also an extension on the bottom right corner that is removable for entering a first pass room.*

The complete system is created by combining the drilling apparatus and the borer mount. The mount is designed such that the drilling apparatus is just behind the operator's canopy on the boring machine, allowing the drilling system to extend up to the mine back. The complete system is shown in relation to the boring machine in Figures 6.6 and 6.7. While mining, visibility of the back of the borer from the operator's seat is slightly impaired, however, the drilling system can be moved to the right or left to give a better view of individual areas of interest. Together, the borer mount and drilling apparatus achieve complete bolt coverage of the right side of the boring machine while fitting into the constrained space required.

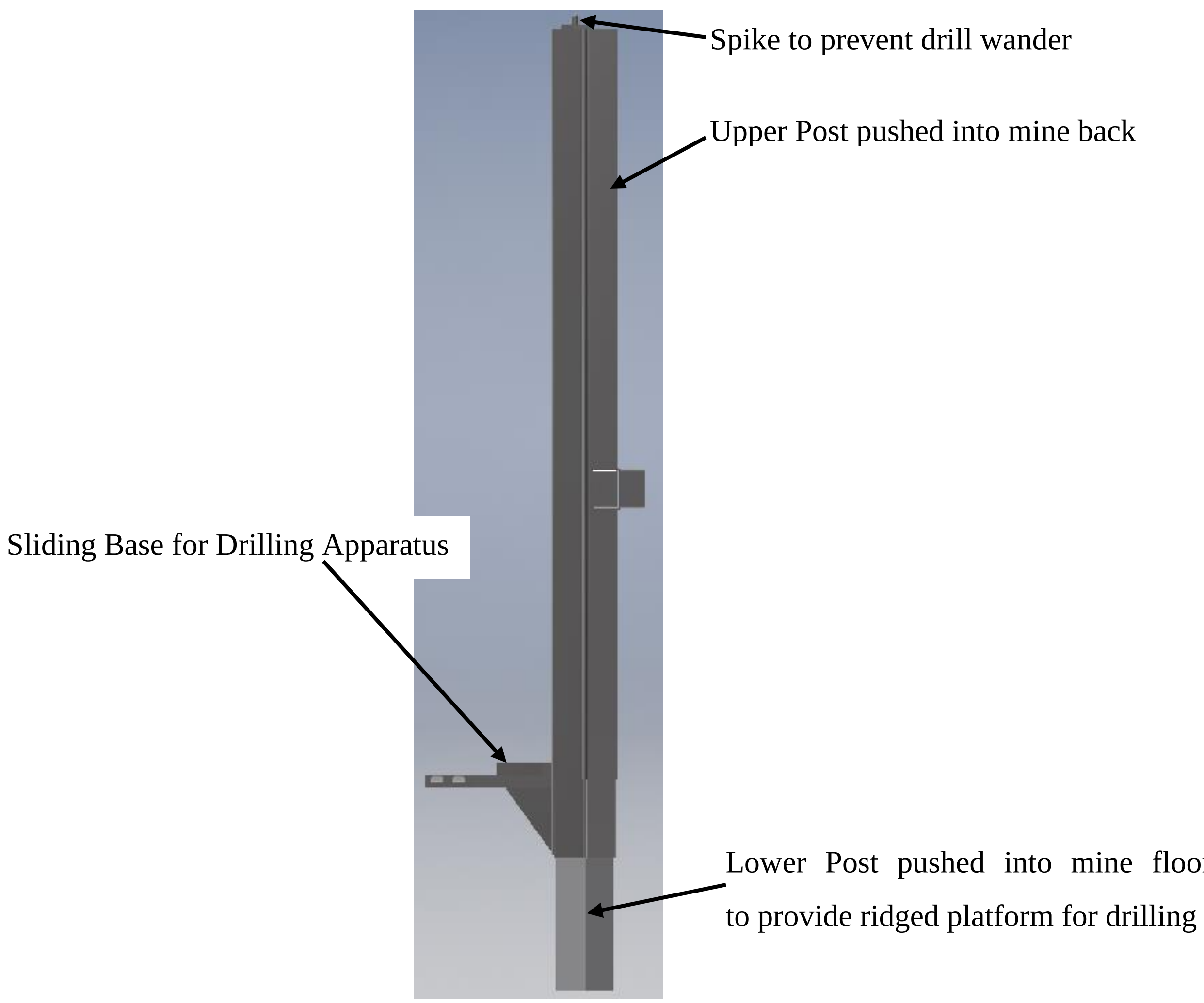



*Figure 6.5 - Drilling system platform and anchoring telescopic post*

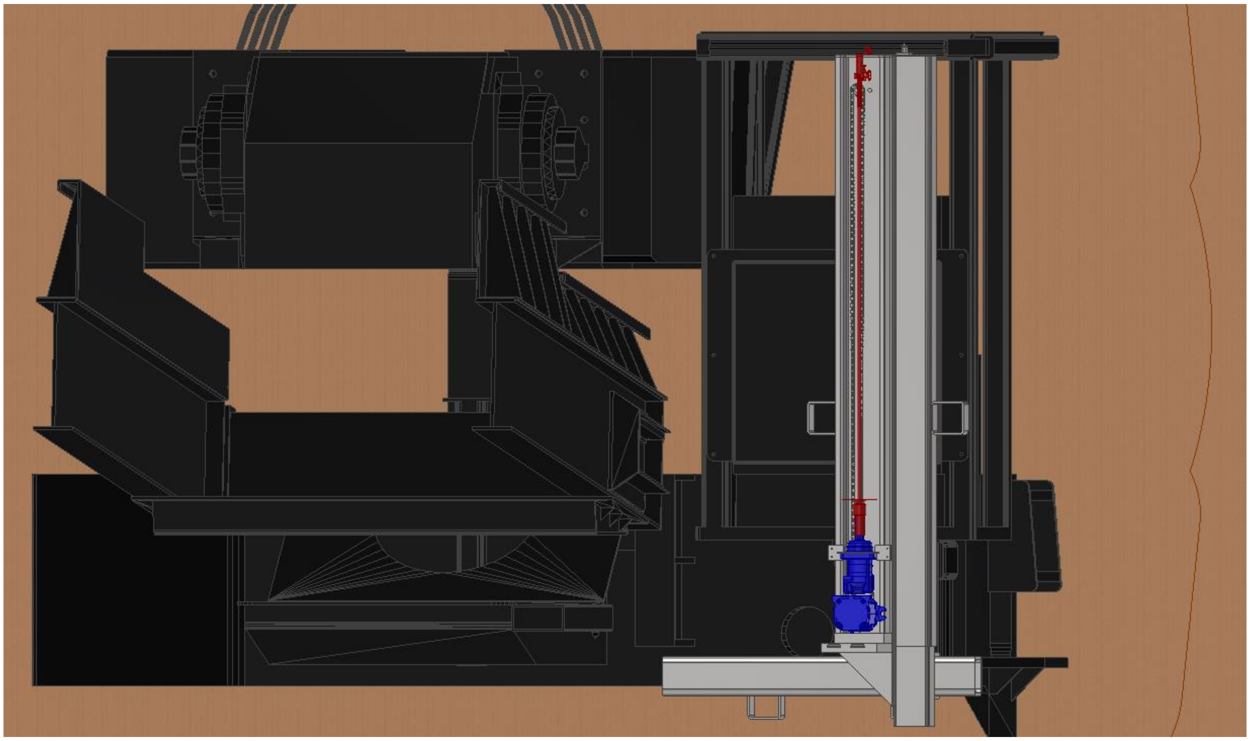


*Figure 6.6 - The complete bolting system attached to a boring machine. Here, the extension shown in Figure 6.4 is not attached.*

The complete system could be furthered by adding several degrees of freedom to compensate for borer movement. The main movement to be compensated for is the forward advance of the machine while mining. This could potentially be attained by adding an extendibility to the borer mount. Such a system would not have to use any control sensors, rather locating the drilling apparatus in the same spot by pressing against the floor and back of the tunnel to lock itself in place. Minimal additional flexibility would be needed to allow for the roll, pitch, and yaw of the boring machine. Using open center valves in the hydraulic positional control system could allow the mount to be flexible while the drilling system is pressed against the floor and back of the tunnel. With these changes, it is expected that the borer mounted bolting system could be given the added ability of bolting while mining is in progress.

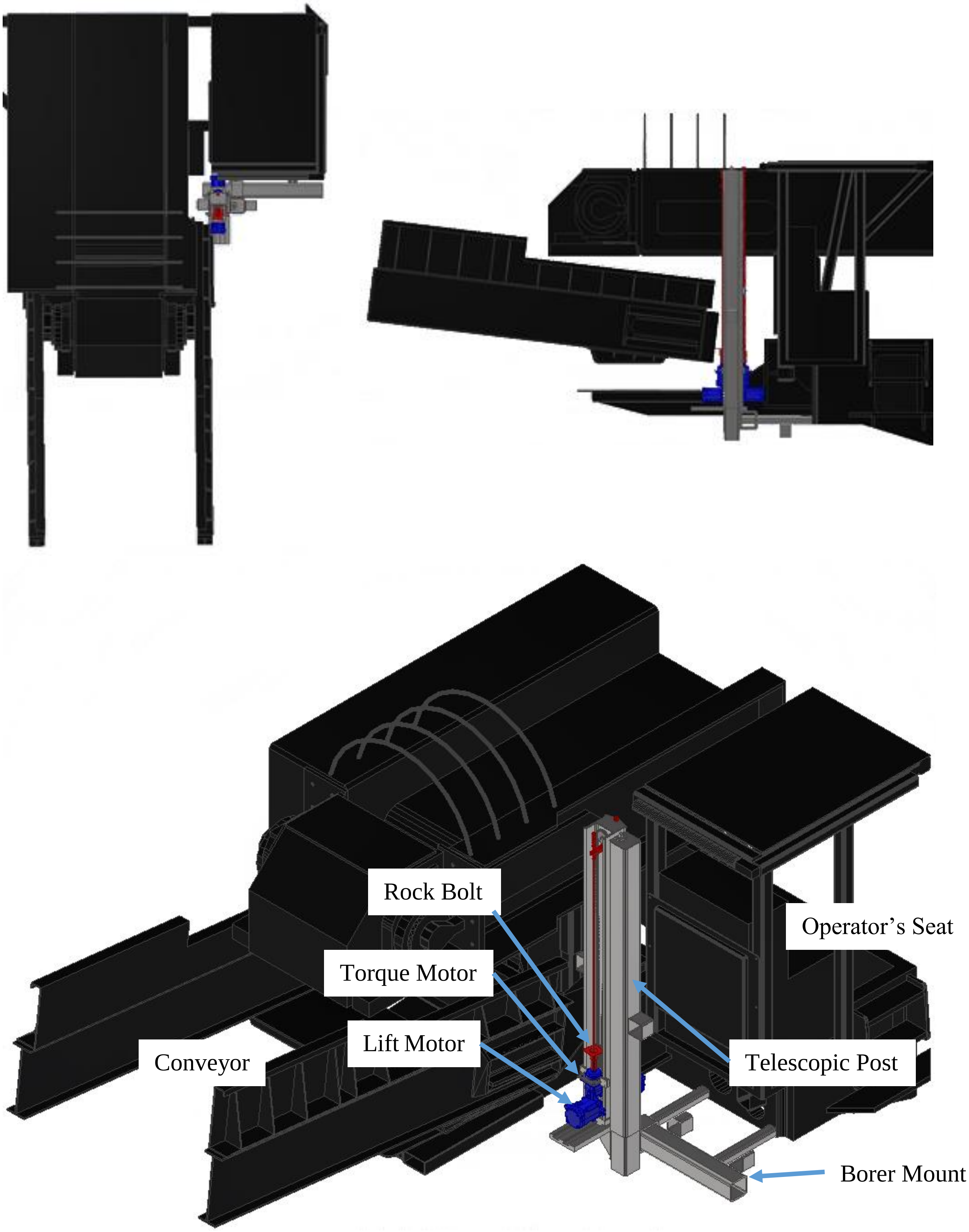



*Figure 6.7 - Top view, side view, and isometric view of the complete mounted bolting system, shown top left, top right, and bottom respectively*

# Chapter 7 Recommendations

This study has shown that a borer-mounted bolting system attached to a continuous boring machine can be advantageous when compared to current practices. The projected time distribution of all the various proposed bolting methods were greatly improved compared to the currently used system. Partially bolting the mined rooms may not affect the work area for the attending underground personnel. However, stability metrics indicate that while an improvement from unbolted ground conditions, partial bolting achieve the same reductions in curvature and displacement as the current cut and bolt procedure. Some recommendations about the location of bolt installation can also be garnered from the stability information. With this information, it is apparent that there are areas in an underground mine that would benefit from the borer mounted bolting system.

Of the methods where time distribution was analyzed, the borer mounted bolting systems capable of installing bolts while mining show the greatest potential for increased productivity. The current anomalous ground bolting procedure suffers from two major sources of lost time in most production rooms: time spent maneuvering the boring and bolting machines, and time spent installing bolts. Of the two, the time spent installing bolts is a much larger portion of the time. The borer mounted bolting system is designed such that the machines needn't be moved as much and therefore saves a large portion of this time from the current anomalous ground bolting procedure. Much greater amounts of time can be saved from the time required to install bolts if the borer is allowed to continue mining while installing the bolts. While the implementation of any borer mounted bolter will save considerable amounts of time, the biggest improvement is realized when the system is capable of installing bolts while mining.

By changing the anomalous ground bolting procedure, the areas where bolts are installed in relation to the progression of the tunnel is changed. There are some areas around the borer that are therefore not supported by bolts during times that there are workers present. Unsupported areas are specifically above and to the left of the conveyor system, as well as any area forward of the back most part of the boring machine. Of these areas, only the operator's seat is left under unbolted ground and a heavy canopy is built into the machine above this area. While the rest of these areas are never used in day to day operations by underground personnel, it is nevertheless recommended that personnel not be allowed in these areas in anomalous ground conditions when partially bolting.

Partially bolting the first pass of a mined tunnel before proceeding to a second pass achieves room stability metrics between that of the unbolted and current anomalous ground scenarios. Partially bolting a mined room may not be recommended for all ground conditions. Ground conditions with greatly reduced salt beam thickness and extremely low clay seam height do not appear to achieve stability metrics comparable to acceptable unbolted ground scenarios. Simulation data showed marginally better stability metrics on the right side of an excavated room are achieved in a partially bolted room with a clay seam height of 0.46 meters as is the case in an unbolted scenario with a clay seam height of 0.61 meters. Partially bolting may be acceptable in low-risk anomalous ground conditions such as leach anomalies. Further simulation and comparison to field data are required before recommendations can be made on specific areas where partial bolting can be used.

FE simulations showed the importance of installing the rock bolts as close to the mine face as possible, before the ore is stressed and deformed. When the unbolted distance to the face is increased, it was shown that the room displacement increases, even when the room is partially bolted along the right side. On the right side of the room where bolts are installed before the second pass is cut there is a reduction in the displacement, indicating that the rock bolts do the most to support the entry if installed prior to the full yielding of the material. There may be a limit to how close to the mine face bolts can be installed without causing premature failure due to excessive ground movement. The simulation model used in this study, a continuum model, does not allow separation of beds, something that may increase the strain on installed bolts above that shown in the model where none of the bolts were shown to yield. This means that regardless of the change in bolting procedure, the bolts should generally be installed as close as possible to the mine face so that the bolts are as effective as possible without causing premature failure.

The borer mounted bolting system can increase production in low-risk anomalous ground conditions. The reduced time in moving various mining machines, and potentially when installing bolts, leaves more time spent cutting with the boring machine. The stability of the tunnel is maintained on the right side of the entry where the mining personnel are located when partially bolting the entry, maintaining safety for the mining personnel without restricting their ability to do their job. For these reasons, the borer mounted bolter system should be implemented underground to improve the production rate in low-risk anomalous ground conditions.

# Chapter 8 Conclusions

Anomalous ground conditions in dry underground potash mines in Saskatchewan cause delays for mining companies in Saskatchewan. Anomalous ground conditions can create reduced stability in the mine back, a hazard mitigated by installing rock bolts with separate machines for mining and bolting. A new system, capable of installing bolts from the continuous boring machines, has been theorized in order to improve the production rate in anomalous ground conditions. This new system requires an altered bolting procedure that results in bolts located differently than the current bolting procedure. The improvements to mining rate make the borer mounted bolter system a useful system in certain ground conditions. This system can safely improve the mining rate of underground potash mines in adverse conditions.

The first step in determining the feasibility of installing a bolting system on the currently used boring machines is to determine the time that can be saved. When the procedural time distribution is calculated for the various proposed and current anomalous ground bolting procedures, it is seen that the proposed partial bolt solution is much faster than the current anomalous bolting procedure. In the case of a system that can bolt while mining, the ideal process time can be achieved in a single pass room. Less time is saved in a two-pass room if the bolting system can only bolt on the right side of the boring machine and requires a specialized bolting machine to fill in the bolting pattern after the entry is complete. Less time is saved by a system that installs bolts while mining is stopped because none of the time spent installing bolts is saved. The time distributions show that the greatest amount of time used while in the anomalous ground cut and bolt procedure is the time taken to install bolts, making the ideal bolting machine one that installs bolts while mining.

Altering the bolting procedure in poor ground conditions, by installing bolts at different locations in relation to the progression of the tunnel, alters the stability of the tunnel. There are two areas of particularly high convex curvature at the edges of where the first mining pass is cut. These areas constitute higher levels of tension in the rock and therefore higher likelihood of failure initiation. Partially bolting the first pass of the tunnel installs bolts in the area along the right edge of the tunnel and greatly reduces the curvature in this area. Towards the left side of the first pass tunnel however, there are no bolts installed until after the second pass is cut. The partial bolting method still resulted in reduced curvature, though to a lesser extent, and similar displacement. The stability metrics analyzed indicate that partial bolting of the entry will result in increased stability when compared to an unbolted room but less stability than the current bolting system.

Based on the research done in this study there are several recommendations that can be made. The borer mounted bolter system will therefore increase the mining rate considerably but should not be used in high-risk anomalous ground conditions.

Developing and implementing a borer mounted bolting system for a continuous mining solution will safely increase production rate in poor ground conditions. The process changes that can be implemented through the use of such a system yield noteworthy production increases, in some scenarios approaching 71%. Even reduced capability systems could reduce the time required in anomalous ground conditions by 57%. A drawback to the borer mounted bolting system is that the tunnel stability expected is between the level currently obtained with the anomalous ground bolting procedure and the level obtained when no bolts are installed. The borer mounted bolter system does require a change in the bolting pattern. Simulations have shown that regardless of the proposed pattern used, the entry will be more stabilized when compared to no bolts being installed, but not to the extent that the current bolting method does. On the right side of the conveyor belt, where mining personnel are located, there is nearly the same amount of stability based on the curvature of the back in this area. A borer-mounted bolting system that implements such a procedure is still safe for mining personnel compared to the current acceptable practices. This will limit the use of the system to some extent. This makes the borer mounted bolting system a viable option in certain ground conditions.

# Chapter 9 Future Work

There are a number of avenues by which the work in this study can be furthered. The material properties used for both materials in the FE models of the tunnel have not been accurately determined, specifically for the clay material. Other bolting options for tunnel modelling could also be considered with the development of a flexible shaft mechanical bolt. More in-depth verification of both the work done in time and tunnel modelling could also be done. Lastly, mine staff should assess individually each mine scenario for areas where the reduced bolt coverage provided by partial bolting procedures can result in adequate stability.

The material properties for the tunnel modelling work are not known exactly. Currently, the potash material properties have been tested but the clay minerals found in the Saskatchewan potash horizon have not been. Most importantly, the average Young's modulus obtained from previous data shows high variability, given as $5.33 \pm 1.10$ GPa and therefore may not be accurate. Further, the material properties of the clay material are determined using model fit values and are therefore affected by the potash Young's modulus. Further, the material models used do not incorporate creep for either potash or clay zone materials and this may have some impact on the simulation results. This effect is likely low over the time frames for excavating a single entry but nevertheless will have some impact on the results. Testing of the clay material found in the potash ore zone, as well as several stratigraphic sections showing the modelled clay seam, would improve the accuracy of the tunnel stability models.

The unbolted distance to the mine face was determined using the minimum feasible distance of installing a bolt behind the boring machine. This distance could be decreased if a mechanical bolt with a flexible shaft was developed. The nearest feasible location to the mine face around the boring machine is behind the operator's canopy because there is not enough space between the tunnel back and the top of the boring machine. Utilizing a flexible shaft bolt, possibly using a cable for a shaft as opposed to a solid steel or fiberglass shaft, may allow this to be done and therefore allow bolts ahead of the operator's canopy.

Further verification work of both the time and tunnel stability modelling can be done. The number of elements used in the models in this study limit the effectiveness of analyzing curvature. In this study the number of elements used were limited by computational power so greater computational power utilizing more elements is needed. Full scale testing of the tunnel displacements, when partially bolting the room, should be done to verify the effects of installing bolts along only one side of a mined tunnel. Likewise, the projected time savings can also be verified by testing. Both require the production of an onboard borer mounted bolting system for installing the bolts.

Further scenarios including current, non-continuous, borer-bolter machines could be used; these machines install bolts much closer to the mine face and therefore may result in reduced displacement by installing bolts before considerable material yielding takes place. It is important to note that in a field scenario where bed separation can occur, installing bolts to close to the mine face may result in premature failing of the bolts due to excessive ground movement immediately after installation. It was shown in the data that the most important parameter in reducing the overall maximum displacement was the unbolted distance to the mine face. Including these machines would result in a more complete breadth of available bolting methods underground.

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# Appendix A – Drilling Cycle Time Data

Table A.1 shows the data points used for determining the overall bolt cycle time. The data was taken via video camera footage and is accurate to one second and the Monte Carlo trials are therefore also accurate to one second.

*Table A.1 - Data used to find drilling time distribution*

| **Process** | **Lining up next hole (s)** | **Retract mast and load bolt (s)** | **Tighten bolt (s)** | **Push bolt into hole (s)** | **Drill head rotation (s)** | **Drill bit retract time (s)** | **Bolt hole drilling time (s)** |
|---|---|---|---|---|---|---|---|
| Data Points | 11 | 7 | 6 | 3 | 2 | 3 | 38 |
| | 11 | 7 | 6 | 3 | 2 | 3 | 34 |
| | 12 | 8 | 8 | 3 | 3 | 3 | 34 |
| | 16 | 10 | 7 | 3 | 2 | 4 | 32 |
| | 14 | 9 | 2 | 4 | 3 | 3 | 34 |
| | 12 | 8 | 4 | 3 | 4 | 3 | 33 |
| | 14 | 9 | 5 | 4 | 3 | 4 | 33 |
| | 10 | 8 | 5 | 4 | 3 | 4 | 36 |
| | | 8 | 5 | 3 | 4 | 4 | 37 |
| | | 7 | 4 | 4 | 3 | 3 | 35 |
| | | 8 | 4 | 4 | 3 | 4 | 41 |
| | | | 4 | 4 | 4 | 4 | 39 |
| | | | 5 | 4 | 4 | 4 | |

## Appendix B – Additional Histogram Data

Attached are the remainder of the histograms from the Monte Carlo uncertainty analysis. Largely, the histograms have similar shapes with different mean values. This was expected because the same underlying data and distributions are used to determine all of the histograms. All histograms represent 1 000 000 trials.

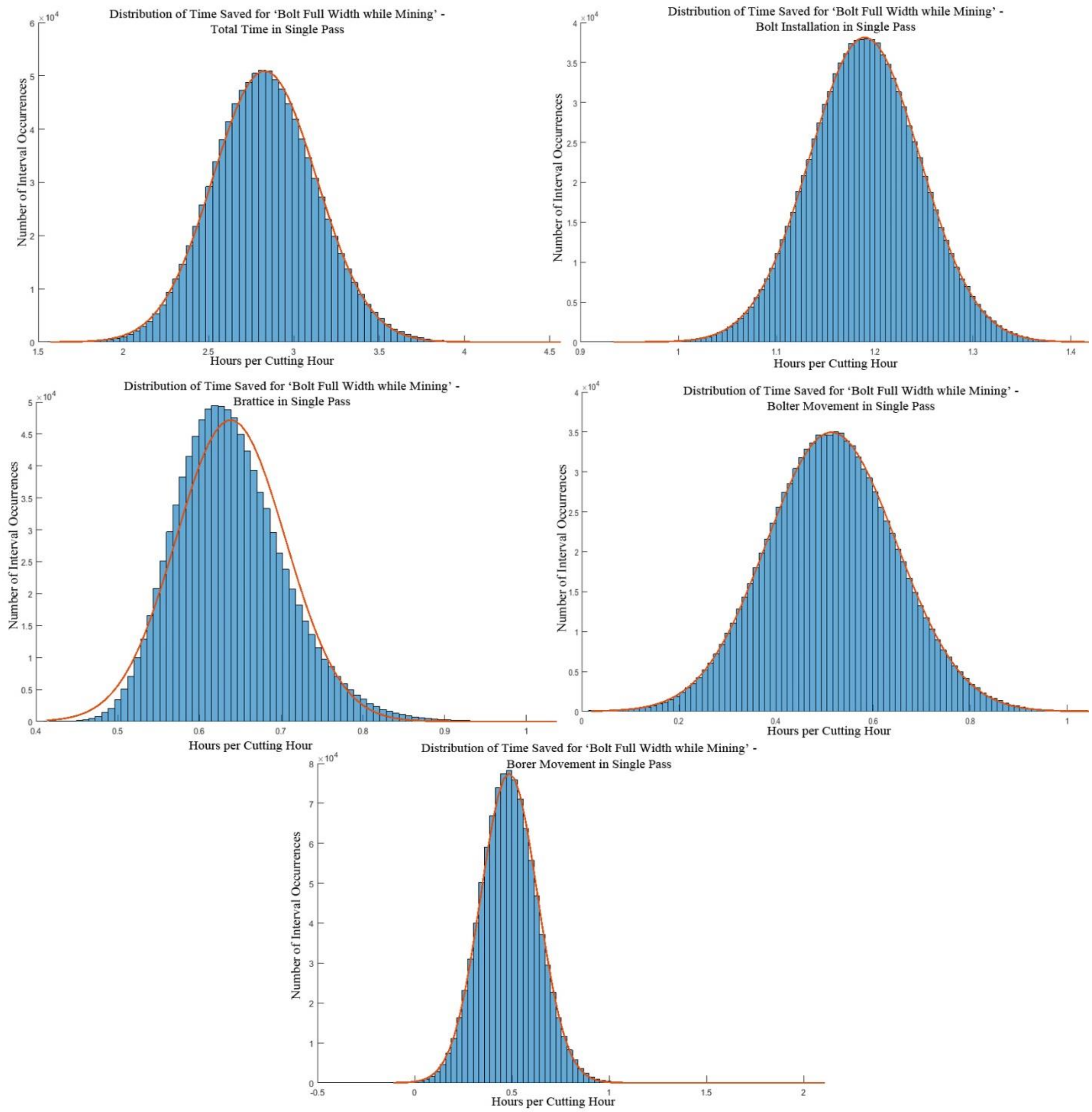


*Figure B.1 - Histograms for installing bolts across a single pass room while mining. Histograms developed using 1 000 000 trials.*

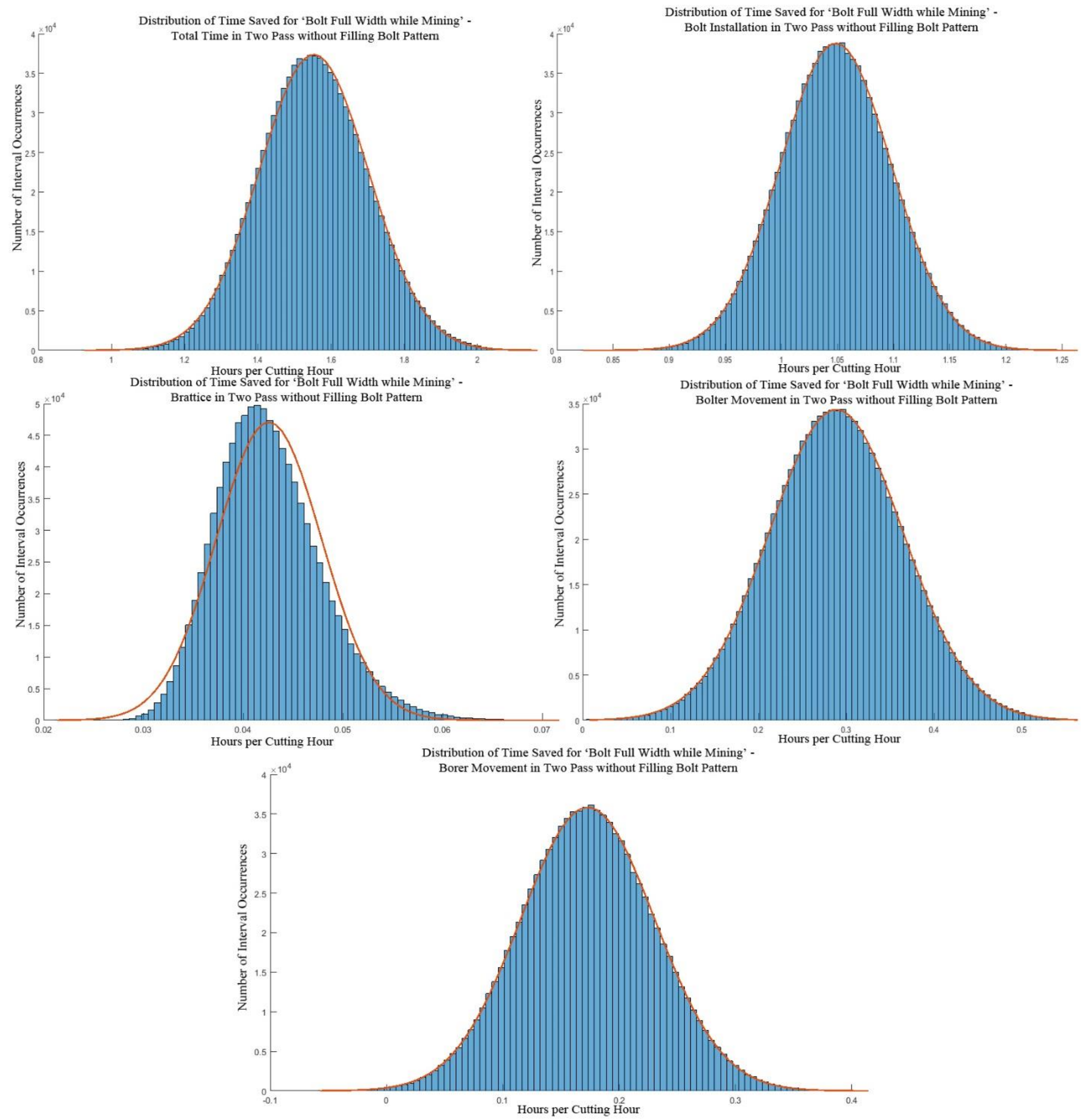


*Figure B.2 - Histograms for installing bolts across a two-pass room while mining. This is without filling the bolting pattern after completion. Histograms developed using 1 000 000 trials. The bolting pattern may not be filled in if the mined room is abandoned after completion. In the case of bolting the full machine width, this does not affect the time distribution as the pattern is already completed when the room is completed.*

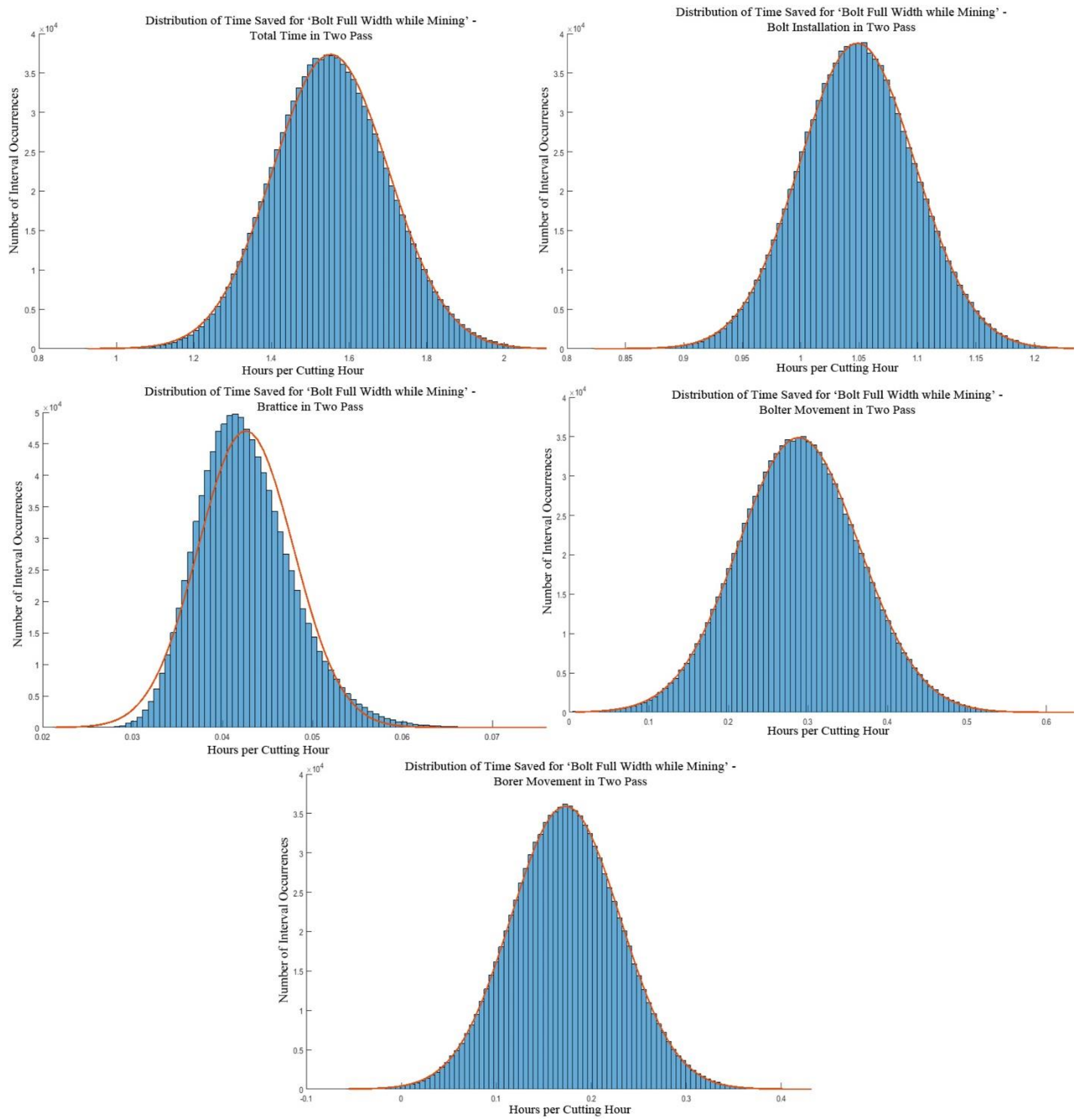


*Figure B.3 - Histograms for installing bolts across a two-pass room while mining. Histograms developed using 1 000 000 trials.*

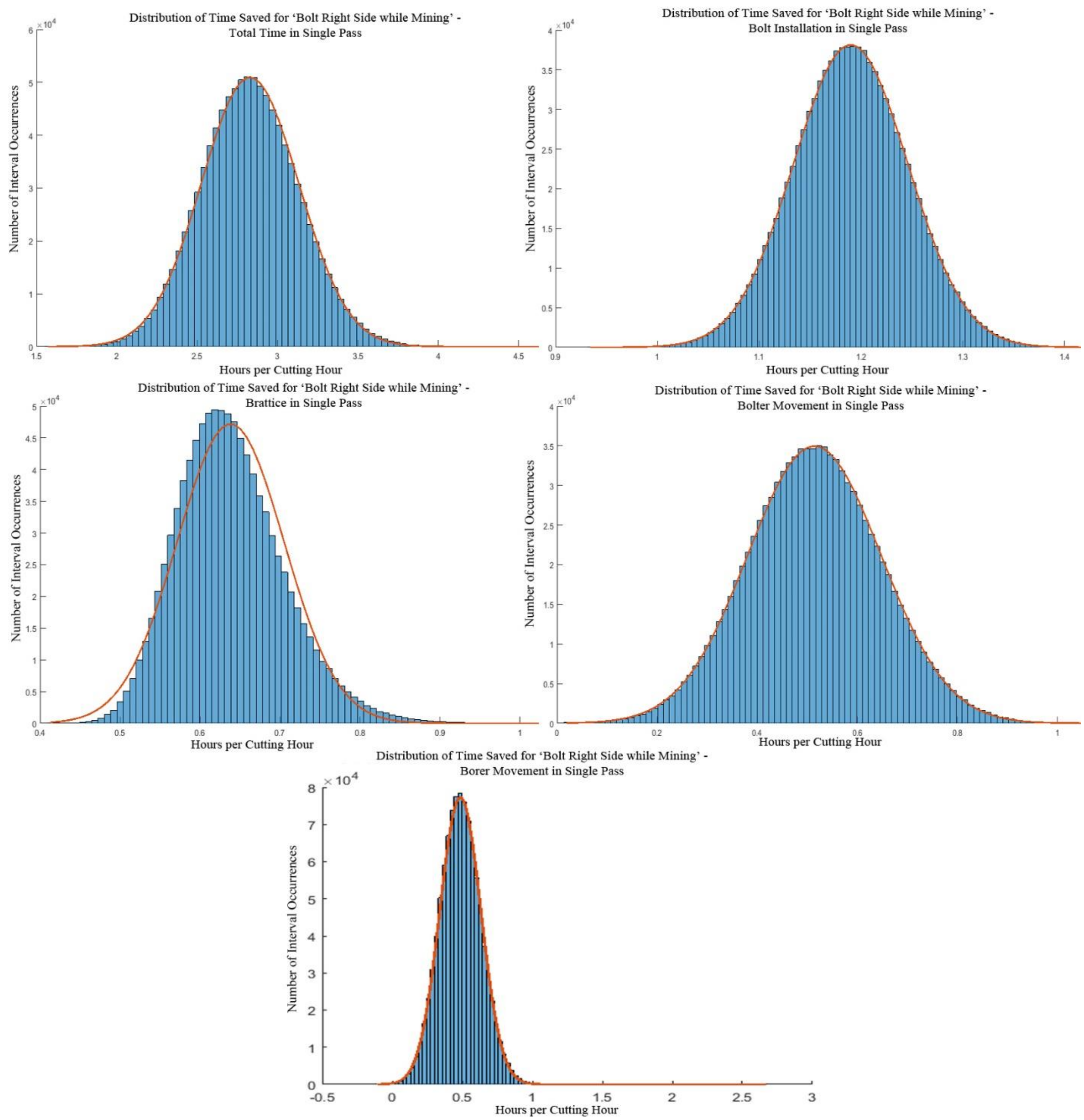


*Figure B.4 - Histograms for installing bolts on the right side while mining a single pass room. Histograms developed using 1 000 000 trials.*

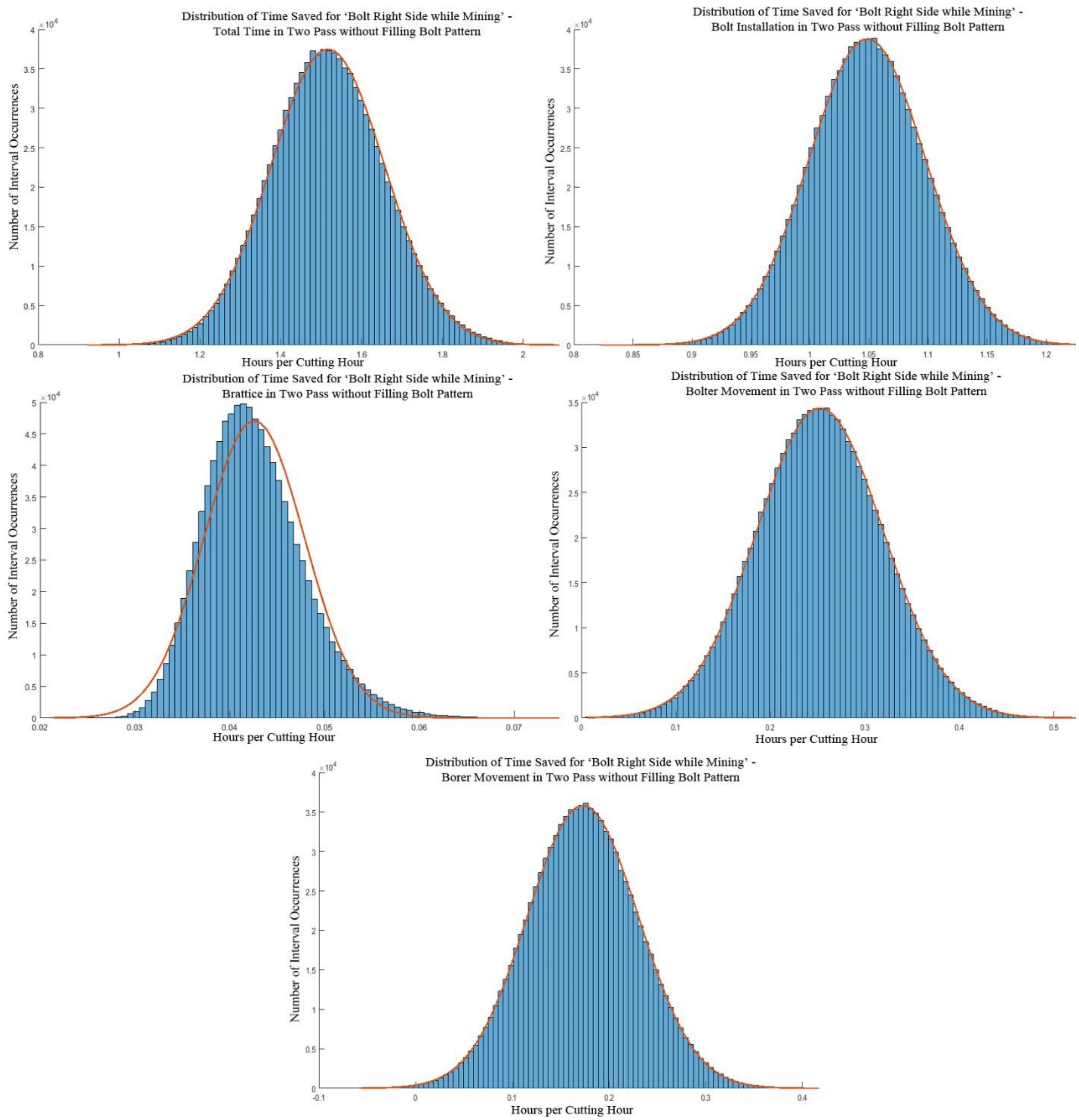


*Figure B.5 - Histograms for installing bolts on the right side while mining in two-pass room. This is without filling the bolting pattern after completion. Histograms developed using 1 000 000 trials. The bolting pattern may not be filled in if the mined room is abandoned after completion.*

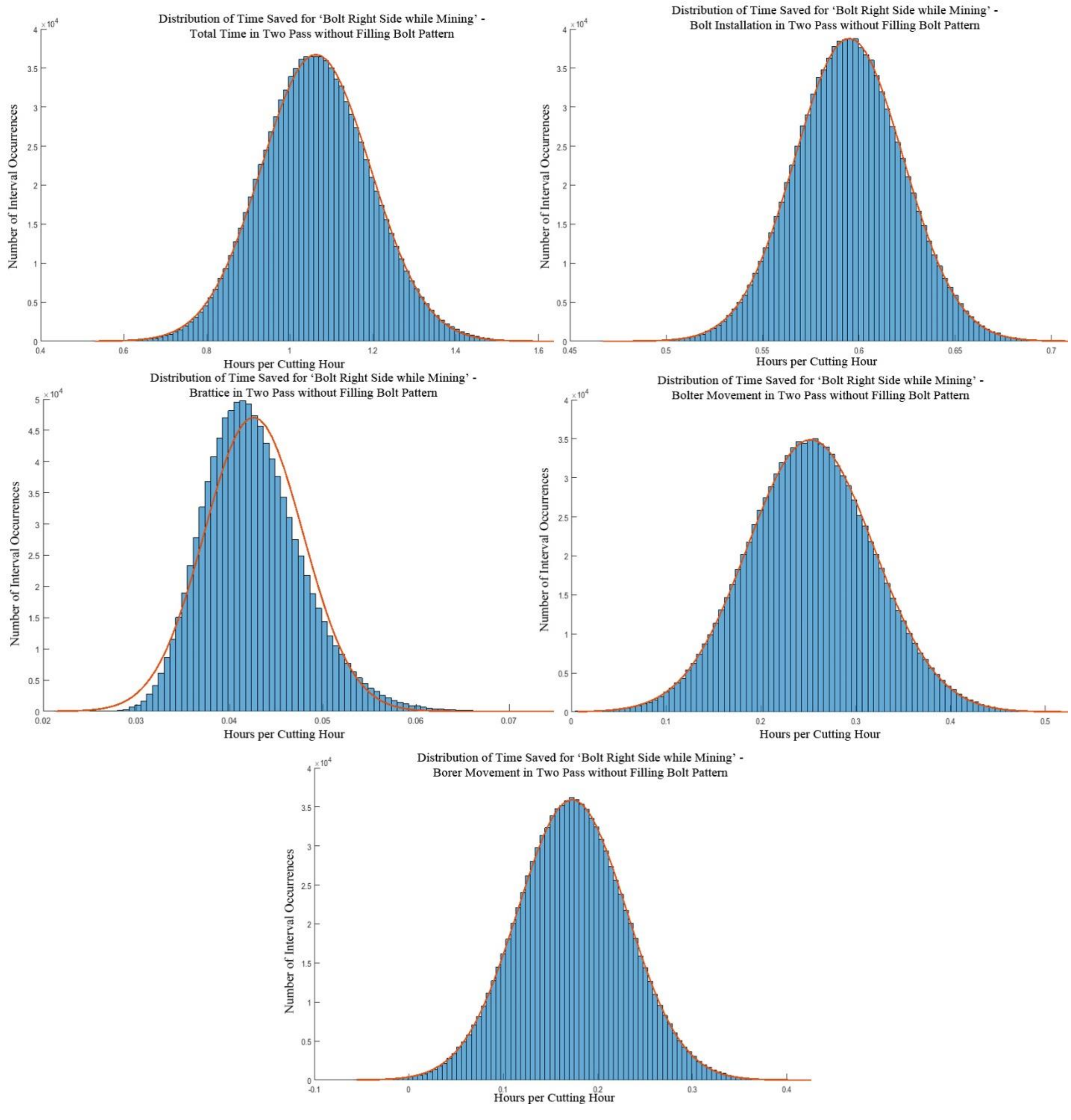


*Figure B.6 - Histograms for installing bolts on the right side while mining in two-pass room. Histograms developed using 1 000 000 trials.*

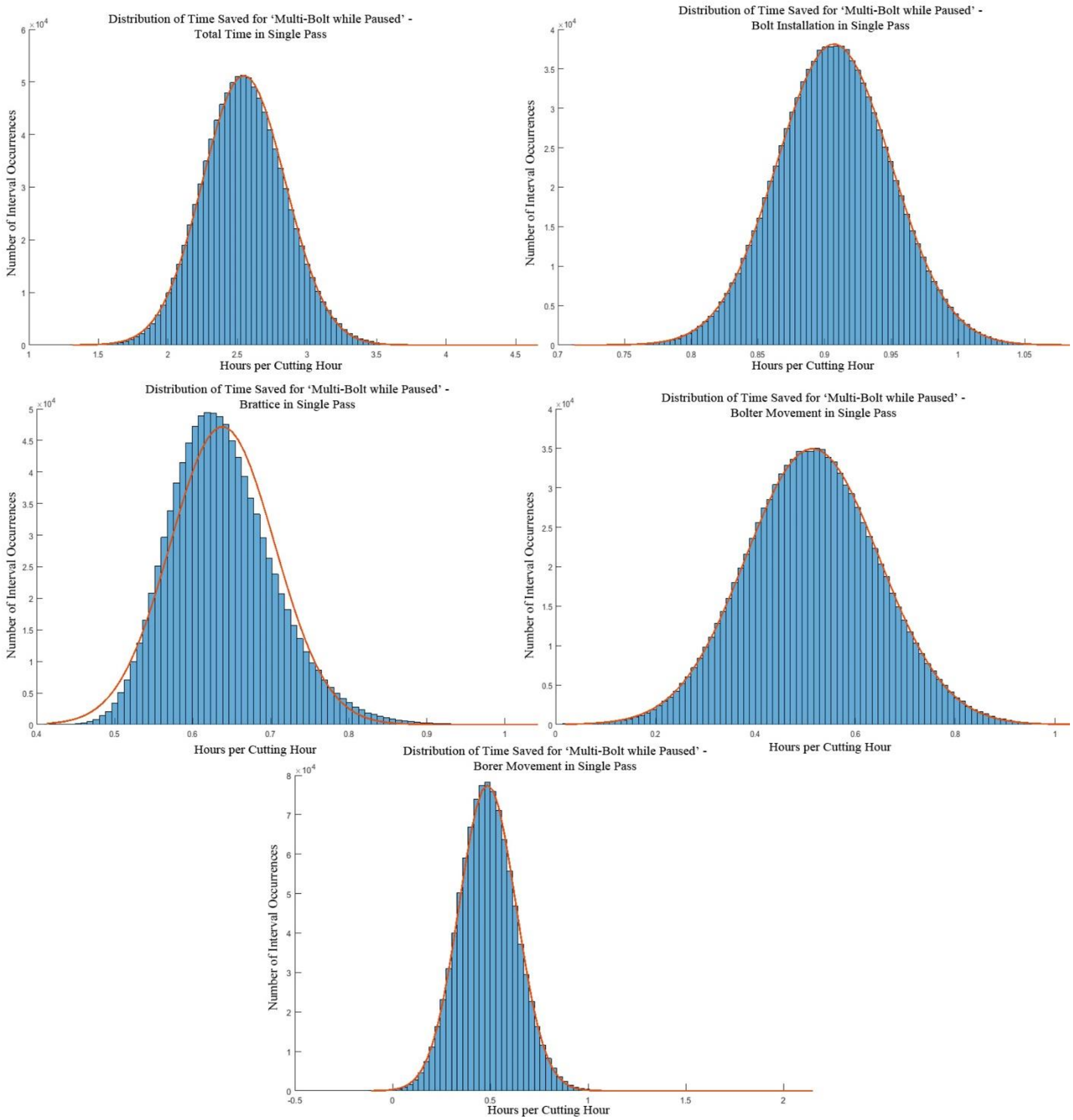


*Figure B.7 - Histograms for installing multiple bolts while paused in a single pass room. Histograms developed using 1 000 000 trials.*

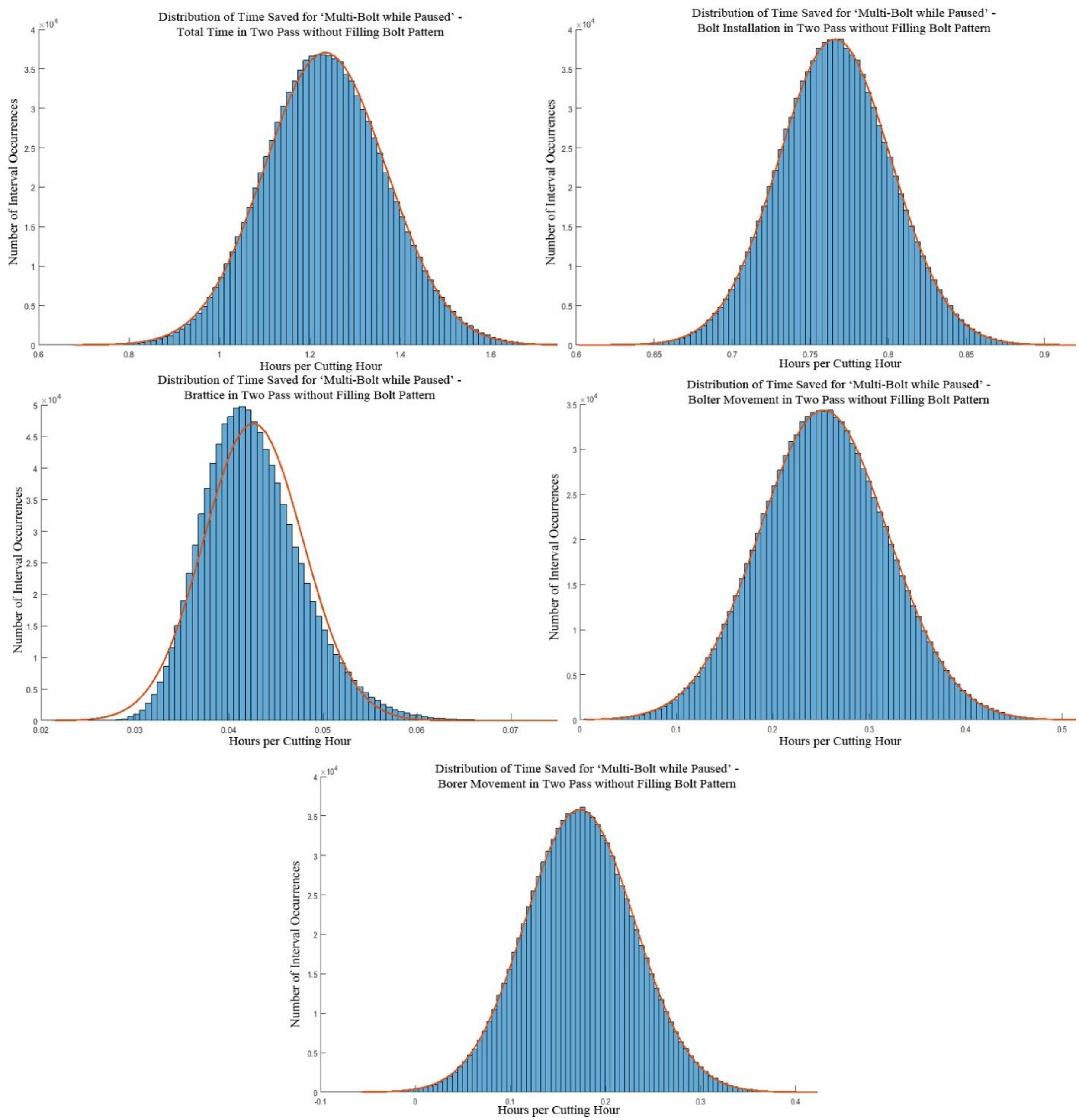


*Figure B.8 - Histograms for installing multiple bolts while paused in a two-pass room. This is without filling the bolting pattern after completion. Histograms developed using 1 000 000 trials. The bolting pattern may not be filled in if the mined room is abandoned after completion.*

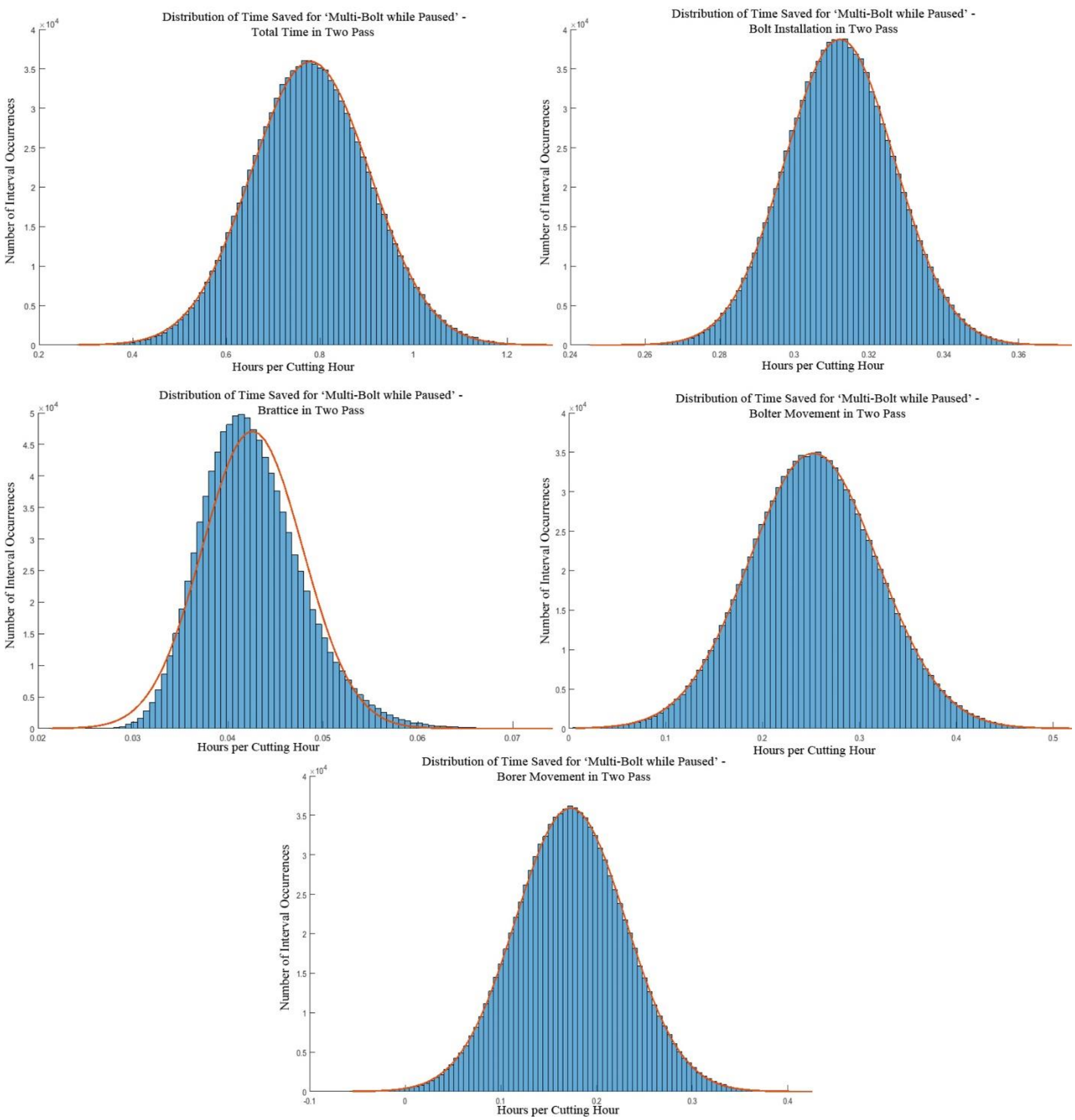


*Figure B.9 - Histograms for installing multiple bolts while paused in a two-pass room. Histograms developed using 1 000 000 trials.*

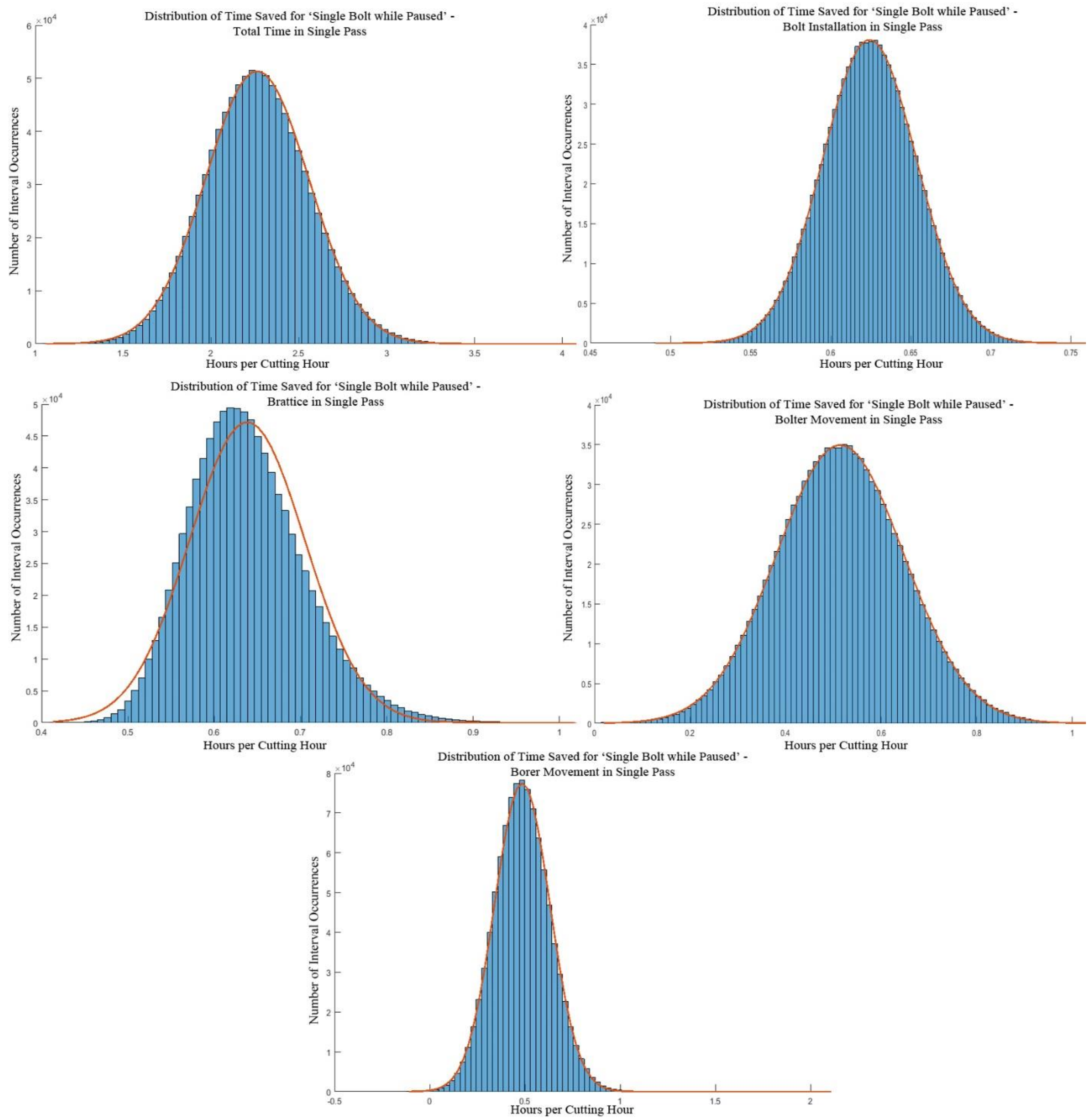


*Figure B.10 - Histograms for installing a single bolt while paused in a single pass room. Histograms developed using 1 000 000 trials.*

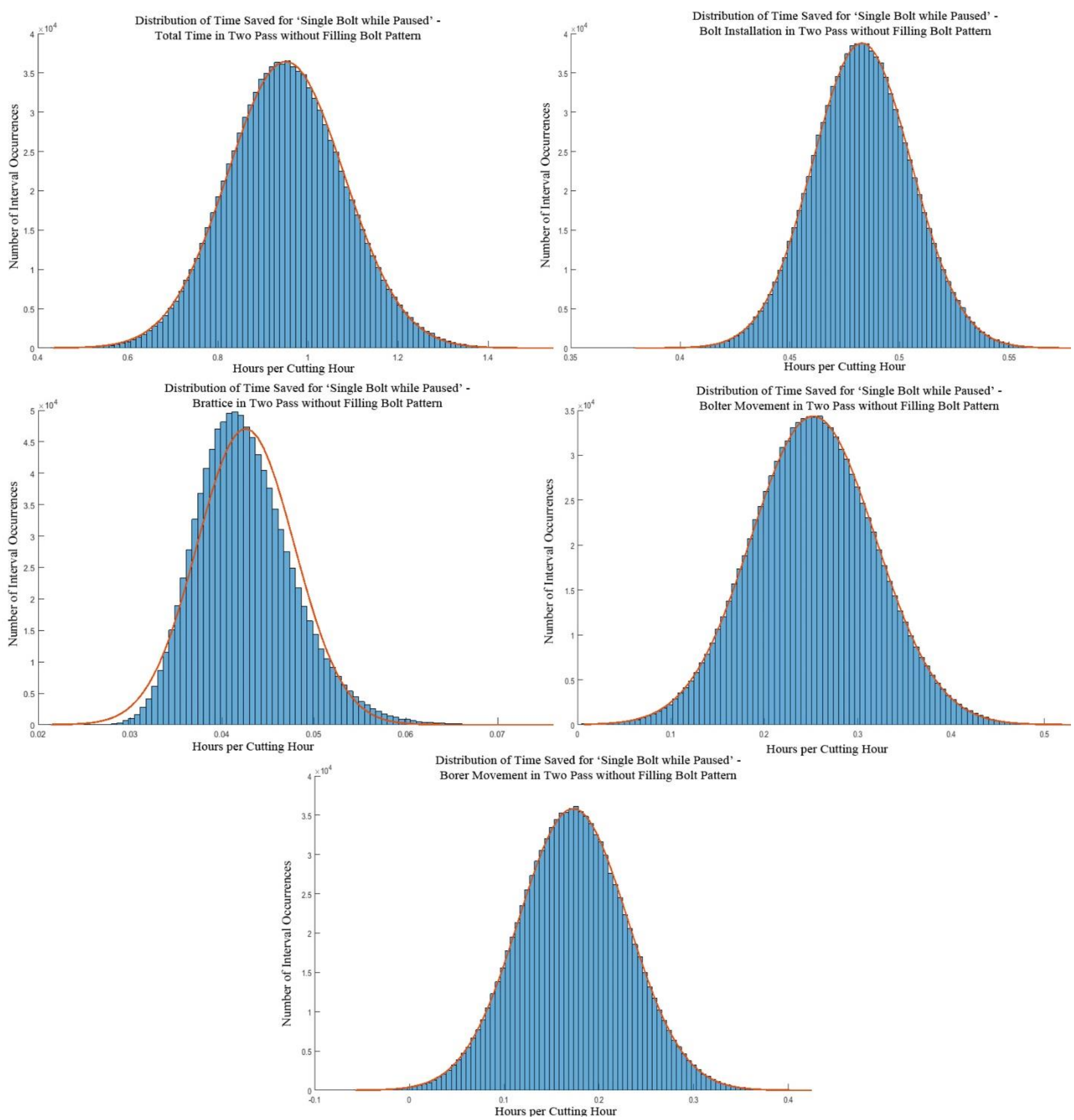


*Figure B.11 - Histograms for installing a single bolt while paused in a two-pass room. This is without filling the bolting pattern after completion. Histograms developed using 1 000 000 trials. The bolting pattern may not be filled in if the mined room is abandoned after completion.*

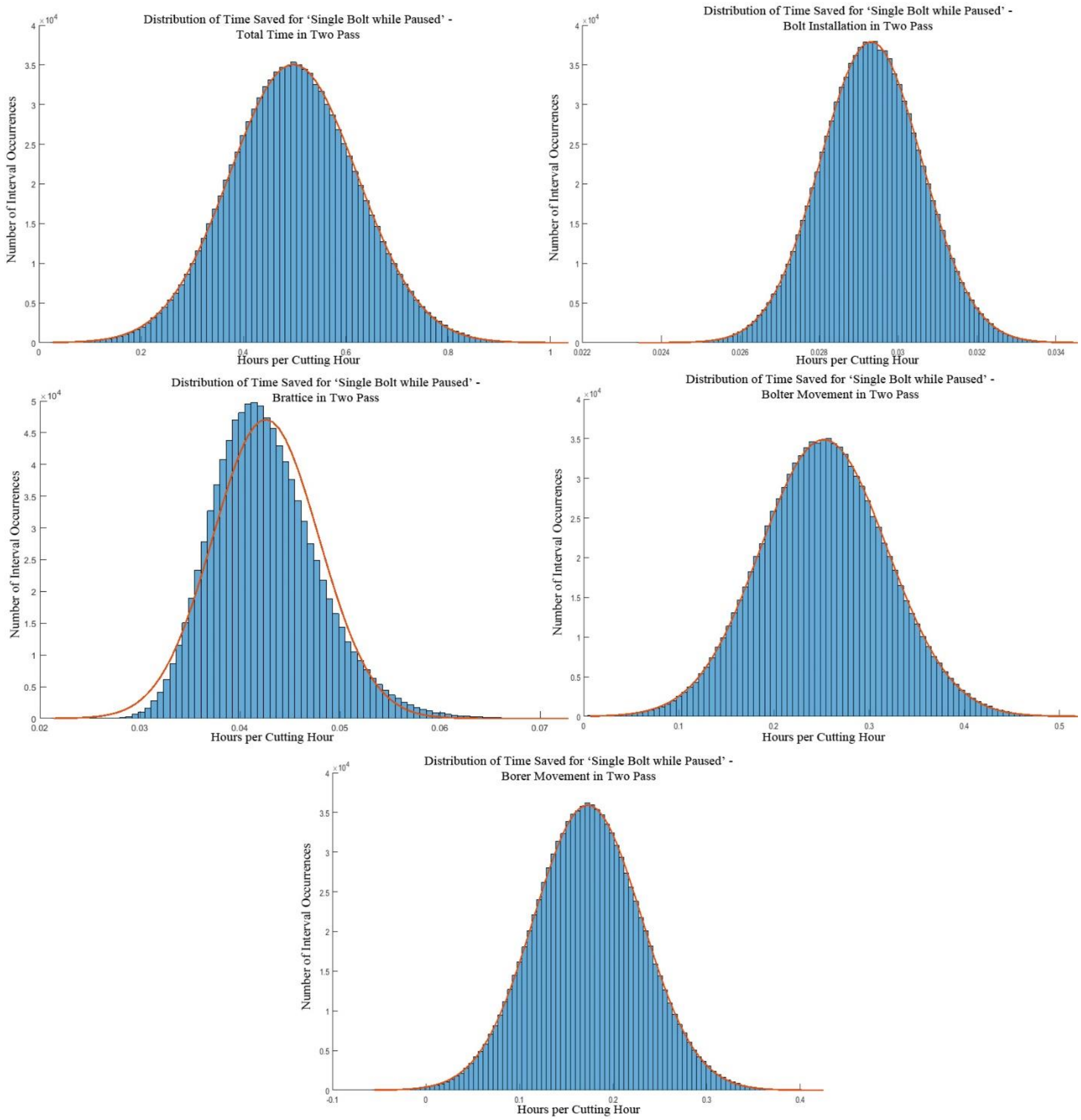


*Figure B.12 - Histograms for installing a single bolt while paused in a two-pass room. Histograms developed using 1 000 000 trials.*

## Appendix C – Simulation Log Files

Attached is the simulation log file for the simulation of the tunnel with a normal clay seam height and no bolts installed. It contains all information necessary for building the simulation as done in the report. Similar log files could be obtained for the other simulations.

# ***$RS^3$ ANALYSIS INFORMATION***

### *PROJECT SETTINGS*

DOCUMENT NAME: TWOPASSROOMNONE.RS3
LAST SAVED WITH RS3 VERSION: 1.02
PROJECT TITLE: PROJECT1
NUMBER OF STAGES: 9
ANALYSIS TYPE: UNCOUPLED
UNITS: METRIC, STRESS AS KPA
TIME UNITS: YEARS
PERMEABILITY UNITS: METERS/SECOND

### STAGE INFORMATION

| # | NAME |
|---|---|
| 1 | INITIAL |
| 2 | FIRSTPASS1 |
| 3 | FIRSTPASS2 |
| 4 | FIRSTPASS3 |
| 5 | FIRSTPASS4 |
| 6 | SECONDPASS1 |
| 7 | SECONDPASS2 |
| 8 | SECONDPASS3 |
| 9 | SECONDPASS4 |

### ORIENTATION

Horizontal Orientation

Plunge: 0 degrees

Trend: 0 degrees

## STRESS ANALYSIS

MAXIMUM NUMBER OF ITERATIONS: 1000
TOLERANCE: 0.001
NUMBER OF LOAD STEPS: 6
CONVERGENCE TYPE: ABSOLUTE FORCE & ENERGY
TENSILE FAILURE: REDUCES SHEAR STRENGTH
JOINT TENSION REDUCES JOINT STIFFNESS BY A FACTOR OF 0.01
TENSILE FAILURE DOES NOT REDUCE HOEK-BROWN TENSILE STRENGTH TO ZERO
DOES NOT USE EFFECTIVE STRESS ANALYSIS

## GROUNDWATER

METHOD: NONE
PORE FLUID UNIT WEIGHT: 9.81 KN/M3

## *FIELD STRESS*

| | | |
|---|---|---|
| TYPE | GRAVITY | |
| GROUND SURFACE ELEVATION: 1000 M | UNIT WEIGHT OF OVERBURDEN: 27 KN/M3 | |
| **SIGMA H1** | | |
| K1 | 1 H1/V | |
| LOCKED-IN | 0 KPA | |
| TREND (DEG) | 0 DEG | |
| PLUNGE (DEG) | 0 DEG | |
| **SIGMA H2** | | **UNIT WEIGHT OF OVERBURDEN: 27 KN/M3** |
| K2 | 1 H1/V | |
| LOCKED-IN | 0 KPA | |

## *SLICES*

| # | NAME | SIZE(M) |
|---|---|---|
| 1 | | 9.144 |
| 2 | | 9.144 |
| 3 | | 9.144 |
| 4 | | 9.144 |
| 5 | | 22.86 |

## *MESH*

MESH DISCRETIZATION TYPE: GRADED
ELEMENT TYPE: 10 NODED TETRAHEDRON

## *MESH QUALITY*

| TYPE | MIN | MAX |
|---|---|---|
| ASPECT RATIO | 0.735941 | 81.0996 |
| MIN DIHEDRAL ANGLE | 0.673072 | 68.9903 |
| MAX DIHEDRAL ANGLE | 71.7211 | 174.171 |
| EDGE LENGTH RATIO | 1.03762 | 108.607 |
| VOLUME | 0.00264158 | 147.742 |

## *MATERIAL PROPERTIES*

### MATERIAL: POTASH

COLOR
INITIAL ELEMENT
LOADING
ELASTIC TYPE
YOUNG'S MODULUS
POISSON'S RATIO
FAILURE CRITERION
PEAK TENSILE
STRENGTH
RESIDUAL TENSILE
STRENGTH PEAK
FRICTION ANGLE PEAK
COHESION
MATERIAL TYPE
DILATION ANGLE
RESIDUAL FRICTION
ANGLE RESIDUAL
COHESION

FIELD
STRESS &
BODY
FORCE
ISOTROPI
C
5.33E+006 KPA
0.17
MOHR-COULOMB
2800 KPA
2100 KPA
35 DEGREES
7000 KPA
PLASTI
C
0 DEGREES
35 DEGREES
6000 KPA

## MATERIAL: CLAY SEAM

COLOR
INITIAL ELEMENT
LOADING
ELASTIC TYPE
YOUNG'S MODULUS
POISSON'S RATIO
FAILURE CRITERION
PEAK TENSILE
STRENGTH
RESIDUAL TENSILE
STRENGTH PEAK
FRICTION ANGLE PEAK
COHESION
MATERIAL TYPE
DILATION ANGLE
RESIDUAL FRICTION
ANGLE RESIDUAL
COHESION

FIELD STRESS & BODY
FORCE ISOTROPIC
5.33E+006 KPA
0.17
MOHR-COULOMB
2000 KPA
1000 KPA
20 DEGREES
5000 KPA
PLASTI
C
0 DEGREES
20 DEGREES
2500 KPA

# *LIST OF ALL COORDINATES*

## 1. [0-9.144M]: EXCAVATION BOUNDARY

| X | Y |
|---|---|
| 9.144 | 0 |
| 9.144 | 3.6576 |
| 3.048 | 3.6576 |
| 0 | 3.6576 |

| | |
|---|---|
| 0 | 0 |
| 3.048 | 0 |

## EXTERNAL BOUNDARY

| | |
|---|---|
| 23.144 | 23.144 |
| 23.144 | 23.144 |
| 23.144 | 23.144 |
| 23.144 | 23.144 |
| 23.144 | 23.144 |
| -14 | -14 |
| -14 | -14 |
| -14 | -14 |

## 1. [0-9.144M]: STAGE BOUNDARY

| X | Y |
|---|---|
| 3.048 | 0 |
| 3.048 | 3.6576 |

## 1. [0-9.144M]: MATERIAL BOUNDARY

| X | Y |
|---|---|
| -14 | 4.4196 |
| 23.144 | 4.4196 |

## 1. [0-9.144M]: MATERIAL BOUNDARY

| X | Y |
|---|---|
| -14 | 4.7858 |
| 23.144 | 4.7858 |

## 2. [9.144-18.288M]: EXCAVATION BOUNDARY

| X | Y |
|---|---|
| 9.144 | 0 |
| 9.144 | 3.6576 |
| 3.048 | 3.6576 |
| 0 | 3.6576 |
| 0 | 0 |
| 3.048 | 0 |

## 2. [9.144-18.288M]: STAGE BOUNDARY

| X | Y |
|---|---|
| 3.048 | 0 |

3.048 3.6576

## 2. [9.144-18.288M]: MATERIAL BOUNDARY

| | |
|---|---|
| -14 | 4.4196 |
| 23.144 | 4.4196 |

## 2. [9.144-18.288M]: MATERIAL BOUNDARY

| X | Y |
|---|---|
| -14 | 4.7858 |
| 23.144 | 4.7858 |

## 3. [18.288-27.432M]: EXCAVATION BOUNDARY

| X | Y |
|---|---|
| 9.144 | 0 |
| 9.144 | 3.6576 |
| 3.048 | 3.6576 |
| 0 | 3.6576 |
| 0 | 0 |
| 3.048 | 0 |

## 3. [18.288-27.432M]: STAGE BOUNDARY

| X | Y |
|---|---|
| 3.048 | 0 |
| 3.048 | 3.6576 |

## 3. [18.288-27.432m]: Material boundary

| X | Y |
|---|---|
| -14 | 4.4196 |
| 23.144 | 4.4196 |

## 3. [18.288-27.432m]: Material boundary

| X | Y |
|---|---|
| -14 | 4.7858 |
| 23.144 | 4.7858 |

### 4. [27.432-36.576m]: Excavation boundary

| X | Y |
|---|---|
| 9.144 | 0 |
| 9.144 | 3.6576 |
| 3.048 | 3.6576 |
| 0 | 3.6576 |
| 0 | 0 |
| 3.048 | 0 |

### 4. [27.432-36.576m]: Stage boundary

| X | Y |
|---|---|
| 3.048 | 0 |
| 3.048 | 3.6576 |

### 4. [27.432-36.576m]: Material boundary

| X | Y |
|---|---|
| -14 | 4.4196 |
| 23.144 | 4.4196 |

### 4. [27.432-36.576m]: Material boundary

| X | Y |
|---|---|
| -14 | 4.7858 |
| 23.144 | 4.7858 |

### 5. [36.576-59.436m]: Material boundary

| X | Y |
|---|---|
| -14 | 4.7858 |
| 23.144 | 4.7858 |

### 5. [36.576-59.436m]: Material boundary

| X | Y |
|---|---|
| -14 | 4.4196 |
| 23.144 | 4.4196 |

## *Results*

## *Stage 1*

### Solid Results

#### Effective Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Effective | 26539.1 | 27387.6 |
| Sigma 2 Effective | 26538.7 | 27366.4 |
| Sigma 3 Effective | 26518.9 | 27365.7 |
| Mean Stress Effective | 26532.2 | 27370.7 |
| Von Mises Stress Effective | 9.40308e-006 | 53.5264 |
| SigmaXX Effective | 26538.7 | 27366.4 |
| SigmaYY Effective | 26519.3 | 27387.3 |
| SigmaZZ Effective | 26538.7 | 27366.4 |
| SigmaXY Effective | -16.855 | 13.0197 |
| SigmaXZ Effective | -16.5135 | 13.9461 |
| SigmaYZ Effective | -22.0272 | 17.5337 |

#### Total Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Total | 26539.1 | 27387.6 |
| Sigma 2 Total | 26538.7 | 27366.4 |
| Sigma 3 Total | 26518.9 | 27365.7 |
| Mean Stress Total | 26532.2 | 27370.7 |
| SigmaXX Total | 26538.7 | 27366.4 |
| SigmaYY Total | 26519.3 | 27387.3 |
| SigmaZZ Total | 26538.7 | 27366.4 |

#### Yielded Elements

| Data Type | Min | Max |
|---|---|---|
| Yielded Elements | 0 | 0 |

#### Displacements

| Data Type | Min | Max |
|---|---|---|
| X Displacement | -1.87431e-005 | 1.9196e-005 |
| Y Displacement | -3.05535e-005 | 1.90485e-005 |
| Z Displacement | -1.6022e-005 | 1.69376e-005 |
| Total Displacement | 0 | 3.29673e-005 |

#### Strength Factor

| Data Type | Min | Max |
|---|---|---|
| Strength Factor | 100 | 100 |

#### Strains

| Data Type | Min | Max |
|---|---|---|
| Volumetric Strain | -1.8488e-005 | 1.9543e-005 |
| Max Shear Strain | 2.57933e-012 | 1.11954e-005 |
| Major Principal Strain | -4.58509e-006 | 1.14612e-005 |
| Mean Principal Strain | -6.06742e-006 | 5.5558e-006 |
| Minor Principal Strain | -9.83503e-006 | 3.49181e-006 |
| StrainXX | -6.42631e-006 | 5.11879e-006 |
| StrainYY | -9.59526e-006 | 1.13318e-005 |
| StrainZZ | -6.1713e-006 | 6.49676e-006 |
| StrainXY | -3.69987e-006 | 2.85799e-006 |
| StrainXZ | -3.62492e-006 | 3.06133e-006 |
| StrainYZ | -4.83525e-006 | 3.84886e-006 |

## *Stage 2*

### Solid Results

#### Effective Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Effective | 3068.68 | 84386.4 |
| Sigma 2 Effective | 1063.51 | 47173.7 |
| Sigma 3 Effective | -9872.39 | 27365.1 |
| Mean Stress Effective | 624.983 | 44861.5 |
| Von Mises Stress Effective | 5.01555e-005 | 61608.8 |
| SigmaXX Effective | -6356.63 | 51963.4 |
| SigmaYY Effective | -7170.19 | 50992.6 |
| SigmaZZ Effective | -9685.48 | 64478.1 |
| SigmaXY Effective | -30647.9 | 24835.9 |
| SigmaXZ Effective | -27966.3 | 32069.5 |
| SigmaYZ Effective | -31573.1 | 29752.1 |

#### Total Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Total | 3068.68 | 84386.4 |
| Sigma 2 Total | 1063.51 | 47173.7 |
| Sigma 3 Total | -9872.39 | 27365.1 |
| Mean Stress Total | 624.983 | 44861.5 |
| SigmaXX Total | -6356.63 | 51963.4 |
| SigmaYY Total | -7170.19 | 50992.6 |
| SigmaZZ Total | -9685.48 | 64478.1 |

#### Yielded Elements

| Data Type | Min | Max |
|---|---|---|
| Yielded Elements | 0 | 100 |

#### Displacements

| Data Type | Min | Max |
| --- | --- | --- |
| X Displacement | -0.0165245 | 0.0163469 |
| Y Displacement | -0.0275628 | 0.0225524 |
| Z Displacement | -0.0155076 | 0.00168061 |
| Total Displacement | 0 | 0.0276952 |

#### Strength Factor

| Data Type | Min | Max |
| --- | --- | --- |
| Strength Factor | -1 | 100 |

#### Strains

| Data Type | Min | Max |
| --- | --- | --- |
| Volumetric Strain | -0.00663295 | 0.010951 |
| Max Shear Strain | 1.23975e-011 | 0.0518146 |
| Major Principal Strain | -0.00111901 | 0.0203759 |
| Mean Principal Strain | -0.00194311 | 0.00381937 |
| Minor Principal Strain | -0.0258485 | 2.85296e-006 |
| StrainXX | -0.0111211 | 0.0098737 |
| StrainYY | -0.0161234 | 0.00699223 |
| StrainZZ | -0.0120443 | 0.00893171 |
| StrainXY | -0.0217167 | 0.0222949 |
| StrainXZ | -0.00613894 | 0.00733046 |
| StrainYZ | -0.0169218 | 0.0139369 |

## *Stage 3*

### Solid Results

#### Effective Stresses

| Data Type | Min | Max |
| --- | --- | --- |
| Sigma 1 Effective | 1373.31 | 85620.9 |
| Sigma 2 Effective | -2368.55 | 46291 |
| Sigma 3 Effective | -7786.6 | 27365.1 |
| Mean Stress Effective | -876.864 | 47229.8 |
| Von Mises Stress Effective | 0.000108841 | 62751.6 |
| SigmaXX Effective | -7295.41 | 55459.8 |
| SigmaYY Effective | -5777.56 | 53099.2 |
| SigmaZZ Effective | -6420.66 | 65339.3 |
| SigmaXY Effective | -31657.7 | 31859 |
| SigmaXZ Effective | -28152.2 | 32243.3 |
| SigmaYZ Effective | -32040.2 | 29772.3 |

#### Total Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Total | 1373.31 | 85620.9 |
| Sigma 2 Total | -2368.55 | 46291 |
| Sigma 3 Total | -7786.6 | 27365.1 |
| Mean Stress Total | -876.864 | 47229.8 |
| SigmaXX Total | -7295.41 | 55459.8 |
| SigmaYY Total | -5777.56 | 53099.2 |
| SigmaZZ Total | -6420.66 | 65339.3 |

#### Yielded Elements

| Data Type | Min | Max |
|---|---|---|
| Yielded Elements | 0 | 100 |

#### Displacements

| Data Type | Min | Max |
|---|---|---|
| X Displacement | -0.0191162 | 0.0193793 |
| Y Displacement | -0.0361371 | 0.0278428 |
| Z Displacement | -0.0148869 | 0.00263347 |
| Total Displacement | 0 | 0.0363237 |

#### Strength Factor

| Data Type | Min | Max |
|---|---|---|
| Strength Factor | -1 | 100 |

#### Strains

| Data Type | Min | Max |
|---|---|---|
| Volumetric Strain | -0.00756835 | 0.0163523 |
| Max Shear Strain | 1.97685e-011 | 0.0850451 |
| Major Principal Strain | -0.00139486 | 0.034458 |
| Mean Principal Strain | -0.00385433 | 0.00512971 |
| Minor Principal Strain | -0.0406538 | 1.07866e-006 |
| StrainXX | -0.0146943 | 0.011799 |
| StrainYY | -0.0234416 | 0.00979666 |
| StrainZZ | -0.00928929 | 0.0100855 |
| StrainXY | -0.0366705 | 0.0350515 |
| StrainXZ | -0.00617976 | 0.00736864 |
| StrainYZ | -0.0198492 | 0.0148086 |

## *Stage 4*

### Solid Results

#### Effective Stresses

| Data Type | Min | Max |
|---|---|---|

| | | |
|---|---|---|
| Sigma 1 Effective | -3071.91 | 85655.8 |
| Sigma 2 Effective | -3999.88 | 46309.2 |
| Sigma 3 Effective | -7905.54 | 27963.6 |
| Mean Stress Effective | -4258.99 | 47421 |
| Von Mises Stress Effective | 9.85347e-005 | 62780 |
| SigmaXX Effective | -6927.51 | 53402.7 |
| SigmaYY Effective | -6185.02 | 53535.9 |
| SigmaZZ Effective | -7320.92 | 65406.6 |
| SigmaXY Effective | -31781.3 | 30124.8 |
| SigmaXZ Effective | -28212.2 | 32351 |
| SigmaYZ Effective | -32048.1 | 29739 |

### Total Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Total | -3071.91 | 85655.8 |
| Sigma 2 Total | -3999.88 | 46309.2 |
| Sigma 3 Total | -7905.54 | 27963.6 |
| Mean Stress Total | -4258.99 | 47421 |
| SigmaXX Total | -6927.51 | 53402.7 |
| SigmaYY Total | -6185.02 | 53535.9 |
| SigmaZZ Total | -7320.92 | 65406.6 |

### Yielded Elements

| Data Type | Min | Max |
|---|---|---|
| Yielded Elements | 0 | 100 |

### Displacements

| Data Type | Min | Max |
|---|---|---|
| X Displacement | -0.0201671 | 0.0206863 |
| Y Displacement | -0.0422679 | 0.0297751 |
| Z Displacement | -0.0151965 | 0.00236795 |
| Total Displacement | 0 | 0.0424334 |

### Strength Factor

| Data Type | Min | Max |
|---|---|---|
| Strength Factor | -1 | 100 |

### Strains

| Data Type | Min | Max |
|---|---|---|

| | | |
|---|---|---|
| Volumetric Strain | -0.00762289 | 0.0231071 |
| Max Shear Strain | 2.21332e-011 | 0.151877 |
| Major Principal Strain | -0.00191871 | 0.0627687 |
| Mean Principal Strain | -0.0039431 | 0.00487532 |
| Minor Principal Strain | -0.0692207 | -1.327e-014 |
| StrainXX | -0.0148099 | 0.0170397 |
| StrainYY | -0.0304588 | 0.00999324 |
| StrainZZ | -0.00968955 | 0.00981286 |
| StrainXY | -0.0606384 | 0.065674 |
| StrainXZ | -0.00796133 | 0.00739451 |
| StrainYZ | -0.0198676 | 0.0150473 |

# Stage 5

## Solid Results

### Effective Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Effective | -3075.55 | 126009 |
| Sigma 2 Effective | -3950.24 | 45770.2 |
| Sigma 3 Effective | -7903.82 | 27938.8 |
| Mean Stress Effective | -4238.86 | 64361.9 |
| Von Mises Stress Effective | 0.000168256 | 93889.6 |
| SigmaXX Effective | -6595.21 | 80130.5 |
| SigmaYY Effective | -6216 | 70044.3 |
| SigmaZZ Effective | -6997.33 | 65360.5 |
| SigmaXY Effective | -31772.2 | 50670.7 |
| SigmaXZ Effective | -28206.8 | 32343.2 |
| SigmaYZ Effective | -32038.2 | 29948.8 |

### Total Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Total | -3075.55 | 126009 |
| Sigma 2 Total | -3950.24 | 45770.2 |
| Sigma 3 Total | -7903.82 | 27938.8 |
| Mean Stress Total | -4238.86 | 64361.9 |
| SigmaXX Total | -6595.21 | 80130.5 |
| SigmaYY Total | -6216 | 70044.3 |
| SigmaZZ Total | -6997.33 | 65360.5 |

### Yielded Elements

| Data Type | Min | Max |
|---|---|---|
| Yielded Elements | 0 | 100 |

### Displacements

| Data Type | Min | Max |
|---|---|---|
| X Displacement | -0.020376 | 0.0207172 |
| Y Displacement | -0.0424582 | 0.0301623 |
| Z Displacement | -0.0149206 | 0.00279777 |
| Total Displacement | 0 | 0.0426309 |

#### Strength Factor

| Data Type | Min | Max |
|---|---|---|
| Strength Factor | -1 | 100 |

#### Strains

| Data Type | Min | Max |
|---|---|---|
| Volumetric Strain | -0.0138781 | 0.0230815 |
| Max Shear Strain | 4.44596e-011 | 0.151884 |
| Major Principal Strain | -0.00191421 | 0.062773 |
| Mean Principal Strain | -0.00387559 | 0.00496634 |
| Minor Principal Strain | -0.0692228 | -1.38478e-014 |
| StrainXX | -0.018761 | 0.0170428 |
| StrainYY | -0.0304671 | 0.0102143 |
| StrainZZ | -0.00888561 | 0.0092537 |
| StrainXY | -0.0606388 | 0.065677 |
| StrainXZ | -0.00796808 | 0.0073928 |
| StrainYZ | -0.0194453 | 0.0150461 |

## *Stage 6*

### Solid Results

#### Effective Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Effective | -3102.11 | 126000 |
| Sigma 2 Effective | -5262.37 | 45716.2 |
| Sigma 3 Effective | -12265.1 | 27926.4 |
| Mean Stress Effective | -4135.47 | 64346 |
| Von Mises Stress Effective | 0.000176594 | 93894.6 |
| SigmaXX Effective | -7085.47 | 80123.3 |
| SigmaYY Effective | -6350.27 | 70040.6 |
| SigmaZZ Effective | -9136.45 | 68153.2 |
| SigmaXY Effective | -29473.1 | 50667.4 |
| SigmaXZ Effective | -28953.6 | 32344.6 |
| SigmaYZ Effective | -34075.8 | 30187.6 |

#### Total Stresses

| Data Type | Min | Max |
|---|---|---|

| | | |
|---|---|---|
| Sigma 1 Total | -3102.11 | 126000 |
| Sigma 2 Total | -5262.37 | 45716.2 |
| Sigma 3 Total | -12265.1 | 27926.4 |
| Mean Stress Total | -4135.47 | 64346 |
| SigmaXX Total | -7085.47 | 80123.3 |
| SigmaYY Total | -6350.27 | 70040.6 |
| SigmaZZ Total | -9136.45 | 68153.2 |

#### Yielded Elements

| Data Type | Min | Max |
|---|---|---|
| Yielded Elements | 0 | 100 |

#### Displacements

| Data Type | Min | Max |
|---|---|---|
| X Displacement | -0.0203937 | 0.0226399 |
| Y Displacement | -0.0448899 | 0.0333399 |
| Z Displacement | -0.0156912 | 0.0027224 |
| Total Displacement | 0 | 0.0453915 |

#### Strength Factor

| Data Type | Min | Max |
|---|---|---|
| Strength Factor | -1 | 100 |

#### Strains

| Data Type | Min | Max |
|---|---|---|
| Volumetric Strain | -0.0138722 | 0.0231326 |
| Max Shear Strain | 4.67673e-011 | 0.151908 |
| Major Principal Strain | -0.00165519 | 0.0627857 |
| Mean Principal Strain | -0.0125205 | 0.00496931 |
| Minor Principal Strain | -0.0692317 | -2.79165e-014 |
| StrainXX | -0.0232177 | 0.0170402 |
| StrainYY | -0.0304705 | 0.0188044 |
| StrainZZ | -0.0134626 | 0.0105584 |
| StrainXY | -0.0607103 | 0.0656856 |
| StrainXZ | -0.00799188 | 0.0073919 |
| StrainYZ | -0.0194478 | 0.0231978 |

## *Stage 7*

### Solid Results

#### Effective Stresses

| Data Type | Min | Max |
|---|---|---|

| | | |
|---|---|---|
| Sigma 1 Effective | -2444.49 | 126133 |
| Sigma 2 Effective | -7330.35 | 45669.3 |
| Sigma 3 Effective | -12602.6 | 27472.7 |
| Mean Stress Effective | -3582.93 | 64417.6 |
| Von Mises Stress Effective | 0.000186503 | 93990.4 |
| SigmaXX Effective | -8281.72 | 80172.1 |
| SigmaYY Effective | -6411.85 | 70145 |
| SigmaZZ Effective | -9086.29 | 68605.7 |
| SigmaXY Effective | -31618.6 | 50726.3 |
| SigmaXZ Effective | -28993.2 | 32350.5 |
| SigmaYZ Effective | -34271.6 | 30217.9 |

### Total Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Total | -2444.49 | 126133 |
| Sigma 2 Total | -7330.35 | 45669.3 |
| Sigma 3 Total | -12602.6 | 27472.7 |
| Mean Stress Total | -3582.93 | 64417.6 |
| SigmaXX Total | -8281.72 | 80172.1 |
| SigmaYY Total | -6411.85 | 70145 |
| SigmaZZ Total | -9086.29 | 68605.7 |

### Yielded Elements

| Data Type | Min | Max |
|---|---|---|
| Yielded Elements | 0 | 100 |

### Displacements

| Data Type | Min | Max |
|---|---|---|
| X Displacement | -0.0203598 | 0.027309 |
| Y Displacement | -0.0498985 | 0.0398982 |
| Z Displacement | -0.01526 | 0.00265482 |
| Total Displacement | 0 | 0.0505332 |

### Strength Factor

| Data Type | Min | Max |
|---|---|---|
| Strength Factor | -1 | 100 |

### Strains

| Data Type | Min | Max |
|---|---|---|

| | | |
|---|---|---|
| Volumetric Strain | -0.0138988 | 0.0310474 |
| Max Shear Strain | 4.95011e-011 | 0.141915 |
| Major Principal Strain | -0.00180597 | 0.0572396 |
| Mean Principal Strain | -0.00850933 | 0.00514376 |
| Minor Principal Strain | -0.0664125 | -6.91383e-014 |
| StrainXX | -0.0387767 | 0.0156202 |
| StrainYY | -0.0305233 | 0.0195122 |
| StrainZZ | -0.0126163 | 0.0108666 |
| StrainXY | -0.0612031 | 0.0593934 |
| StrainXZ | -0.00811394 | 0.0142985 |
| StrainYZ | -0.0194483 | 0.0238849 |

## *Stage 8*

### Solid Results

#### Effective Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Effective | -2350.85 | 127873 |
| Sigma 2 Effective | -6336.63 | 45785 |
| Sigma 3 Effective | -11162.5 | 27539.9 |
| Mean Stress Effective | -3509.21 | 65366.8 |
| Von Mises Stress Effective | 0.000205255 | 95148.9 |
| SigmaXX Effective | -8151.55 | 81263.8 |
| SigmaYY Effective | -6449.01 | 71368.1 |
| SigmaZZ Effective | -9052.14 | 68622.3 |
| SigmaXY Effective | -31716.9 | 51309.4 |
| SigmaXZ Effective | -28987.4 | 32350.7 |
| SigmaYZ Effective | -34277.3 | 30216.8 |

#### Total Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Total | -2350.85 | 127873 |
| Sigma 2 Total | -6336.63 | 45785 |
| Sigma 3 Total | -11162.5 | 27539.9 |
| Mean Stress Total | -3509.21 | 65366.8 |
| SigmaXX Total | -8151.55 | 81263.8 |
| SigmaYY Total | -6449.01 | 71368.1 |
| SigmaZZ Total | -9052.14 | 68622.3 |

#### Yielded Elements

| Data Type | Min | Max |
|---|---|---|
| Yielded Elements | 0 | 100 |

#### Displacements

| Data Type | Min | Max |
|---|---|---|
| X Displacement | -0.0204907 | 0.022278 |
| Y Displacement | -0.0534273 | 0.0426555 |
| Z Displacement | -0.0150877 | 0.0027733 |
| Total Displacement | 0 | 0.0538646 |

#### Strength Factor

| Data Type | Min | Max |
|---|---|---|
| Strength Factor | -1 | 100 |

#### Strains

| Data Type | Min | Max |
|---|---|---|
| Volumetric Strain | -0.0142514 | 0.0271634 |
| Max Shear Strain | 5.46689e-011 | 0.142073 |
| Major Principal Strain | -0.00183013 | 0.0573065 |
| Mean Principal Strain | -0.00857966 | 0.00536025 |
| Minor Principal Strain | -0.0664808 | -1.10486e-013 |
| StrainXX | -0.0227539 | 0.0156072 |
| StrainYY | -0.0306837 | 0.0153316 |
| StrainZZ | -0.013588 | 0.0108815 |
| StrainXY | -0.061272 | 0.0593533 |
| StrainXZ | -0.00816023 | 0.00743558 |
| StrainYZ | -0.019852 | 0.0238849 |

## *Stage 9*

### Solid Results

#### Effective Stresses

| Data Type | Min | Max |
|---|---|---|
| Sigma 1 Effective | -2346.48 | 131704 |
| Sigma 2 Effective | -3624.73 | 46845.1 |
| Sigma 3 Effective | -11180.1 | 27411.4 |
| Mean Stress Effective | -3505.43 | 67299.8 |
| Von Mises Stress Effective | 0.000280466 | 97860.8 |
| SigmaXX Effective | -7126.43 | 83784.4 |
| SigmaYY Effective | -6371.75 | 74000.7 |
| SigmaZZ Effective | -9073.79 | 68591.6 |
| SigmaXY Effective | -31717.3 | 52574.1 |
| SigmaXZ Effective | -28983.3 | 32349.9 |
| SigmaYZ Effective | -34269.9 | 32831.6 |

#### Total Stresses

| Data Type | Min | Max |
|---|---|---|

| | | |
|---|---|---|
| Sigma 1 Total | -2346.48 | 131704 |
| Sigma 2 Total | -3624.73 | 46845.1 |
| Sigma 3 Total | -11180.1 | 27411.4 |
| Mean Stress Total | -3505.43 | 67299.8 |
| SigmaXX Total | -7126.43 | 83784.4 |
| SigmaYY Total | -6371.75 | 74000.7 |
| SigmaZZ Total | -9073.79 | 68591.6 |

### Yielded Elements

| Data Type | Min | Max |
|---|---|---|
| Yielded Elements | 0 | 100 |

### Displacements

| Data Type | Min | Max |
|---|---|---|
| X Displacement | -0.0206971 | 0.0212638 |
| Y Displacement | -0.0536579 | 0.0428137 |
| Z Displacement | -0.0173312 | 0.00319631 |
| Total Displacement | 0 | 0.0540996 |

### Strength Factor

| Data Type | Min | Max |
|---|---|---|
| Strength Factor | -1 | 100 |

### Strains

| Data Type | Min | Max |
|---|---|---|
| Volumetric Strain | -0.0149695 | 0.0271535 |
| Max Shear Strain | 7.5239e-011 | 0.142073 |
| Major Principal Strain | -0.00182986 | 0.0573056 |
| Mean Principal Strain | -0.00666969 | 0.00542547 |
| Minor Principal Strain | -0.0664814 | -9.21794e-014 |
| StrainXX | -0.0206123 | 0.0156052 |
| StrainYY | -0.0306831 | 0.0122739 |
| StrainZZ | -0.0104312 | 0.0141937 |
| StrainXY | -0.0612717 | 0.0593541 |
| StrainXZ | -0.00816334 | 0.0074354 |
| StrainYZ | -0.0346736 | 0.0238839 |

# Appendix D – Additional Displacement Profiles

A complete set of displacement profiles for the tunnel modelling section are attached. The back-displacement profiles generally fit into three groups: the current anomalous ground bolting procedure, the proposed bolting methods, and the unbolted tunnel scenario. The current anomalous ground bolting procedure allows the least displacement and has the most gradual curvature, while the unbolted scenario has the most displacement and sharpest curvature. All the proposed methods show reasonably similar displacement profiles. Improved stability is seen in the proposed models along the right side of the entry where bolts are installed in first pass as the curvature is reduced substantially in the area of high yield along this pillar. All figures use stages as defined in Figure D.1 below.

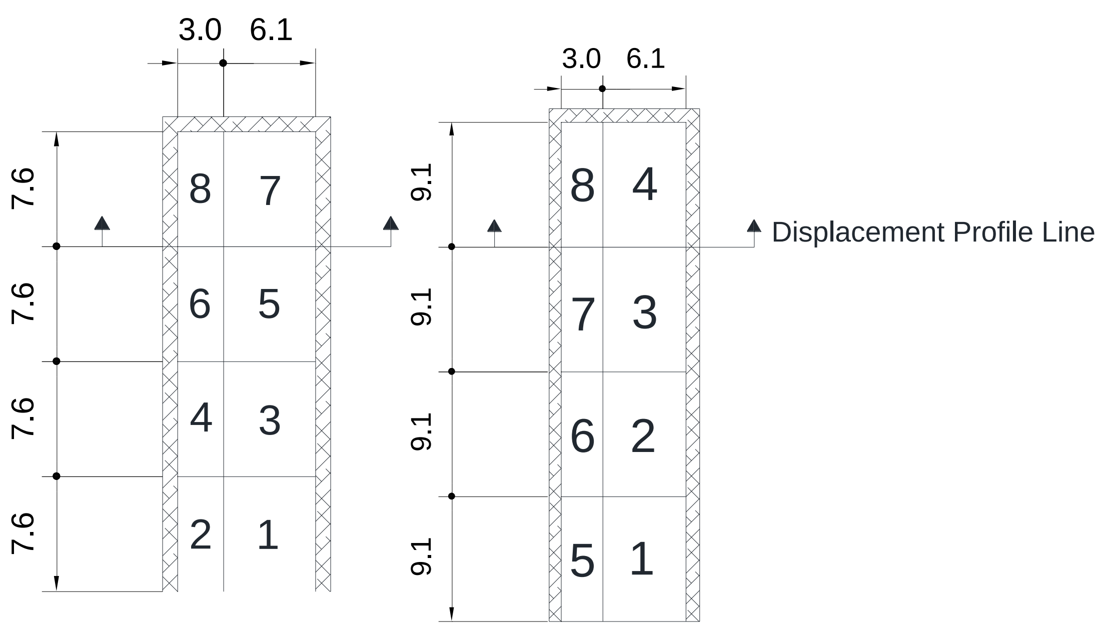


*Figure D.1 - The ordered cutting sequences that were analyzed for the current pattern, on the left, and the proposed cutting pattern, on the right. All dimensions are in meters. The numbers represent the order used to remove the different potash sections. On the left, the current bolting pattern removes the total width of the entry before proceeding down the tunnel. This is unlike the current normal ground method and proposed methods, on the right, where the entire length of the tunnel is cut in the first pass before proceeding to widen the tunnel to its fully finished width. The line where the displacement profile is taken is also marked above. In both cases, it is one cutting section from the end of the finished tunnel, 7.6 meters in the current procedure and 9.1 meters in the proposed.*

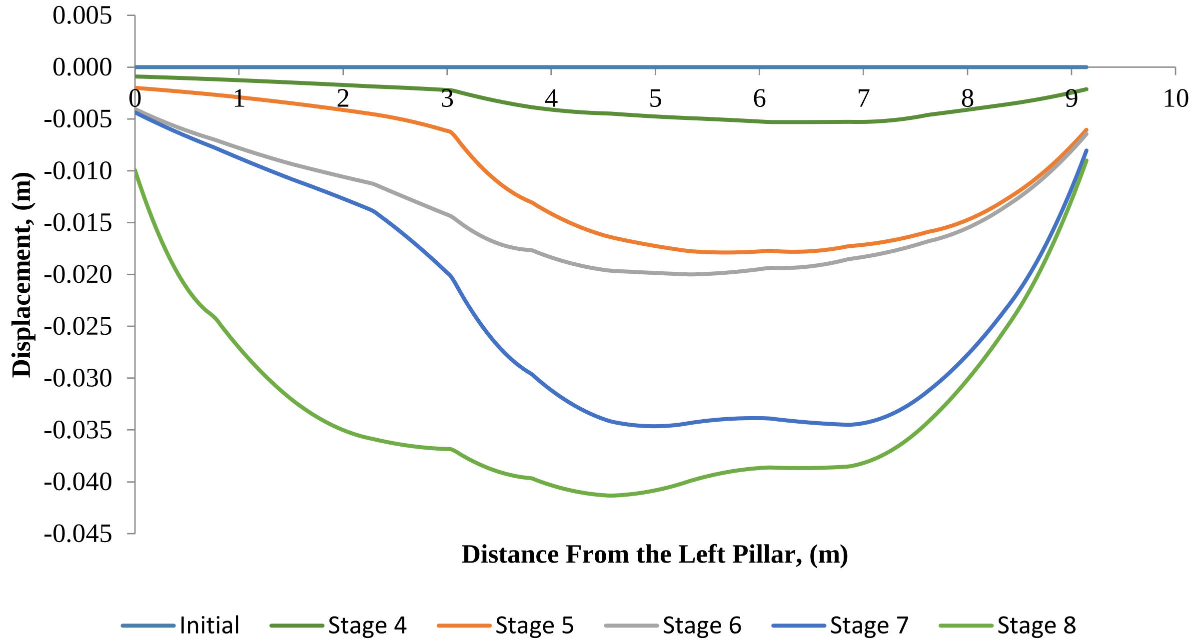


*Figure D.2 - Profile for the current bolting pattern with a 0.76-meter-thick salt beam. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, Stage 2, and Stage 3, which are removed for clarity. This bolting method offers the most gradual displacement curvature and lowest displacement of any method.*

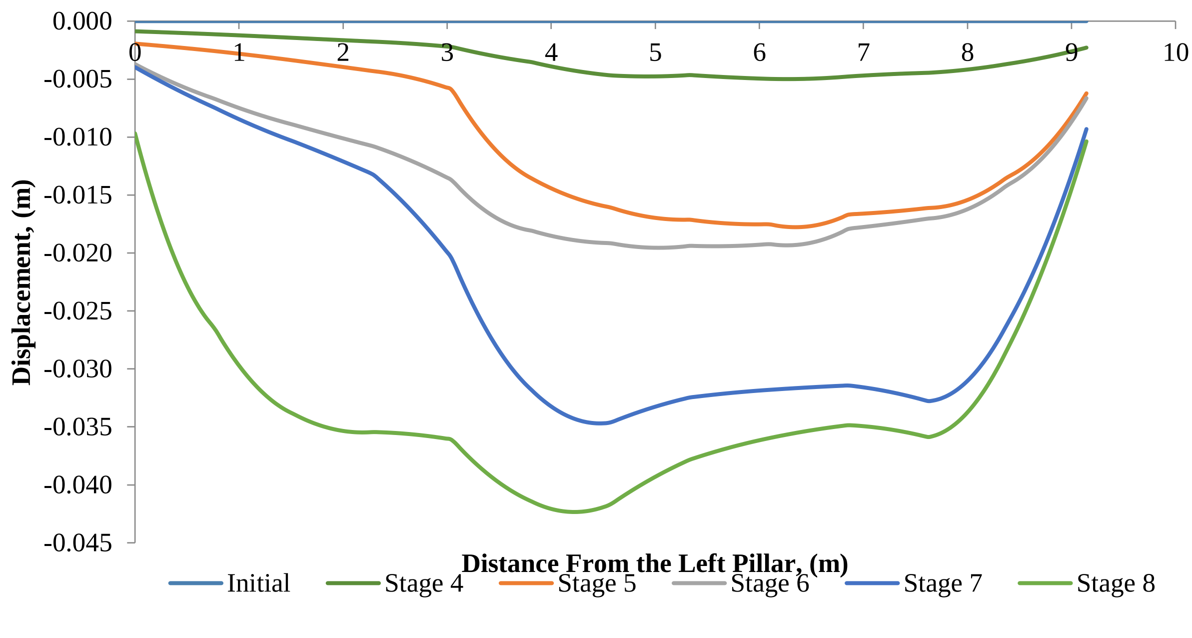


*Figure D.3 - Profile for the current bolting pattern with 0.46-meter-thick salt beam. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, Stage 2, and Stage 3, which are removed for clarity. This bolting method offers the most gradual displacement curvature and lowest displacement of any method.*

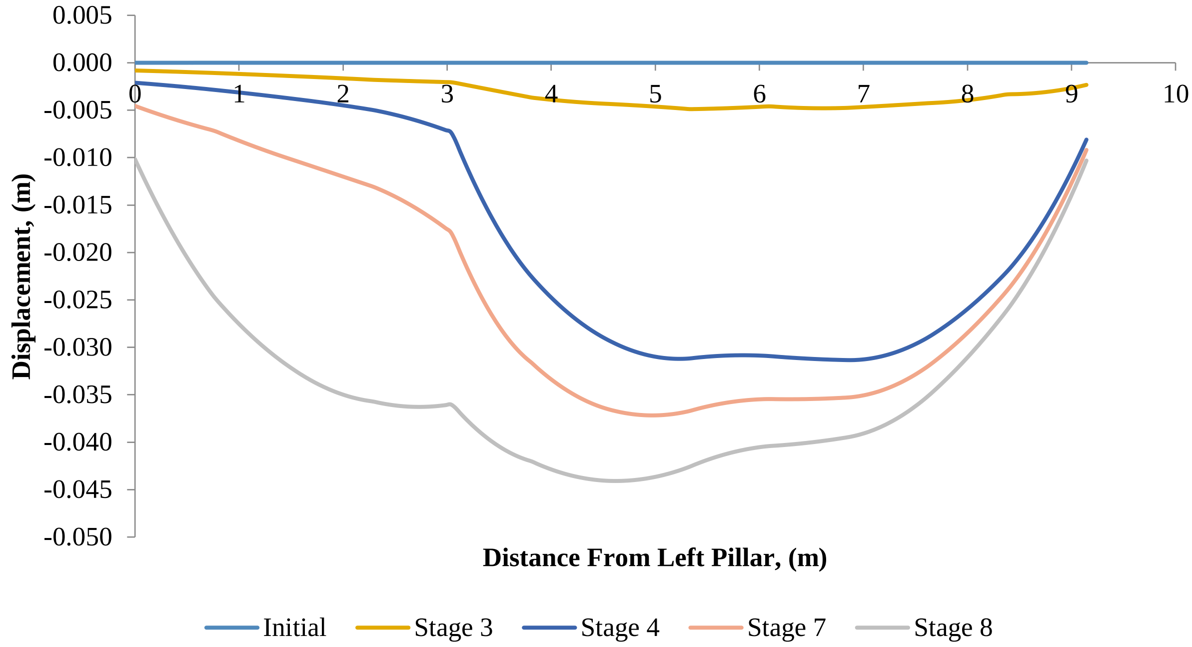


*Figure D.4 - Profile for an unbolted tunnel with a 0.76-meter-thick salt beam. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. Overlapping stages are removed for clarity.*

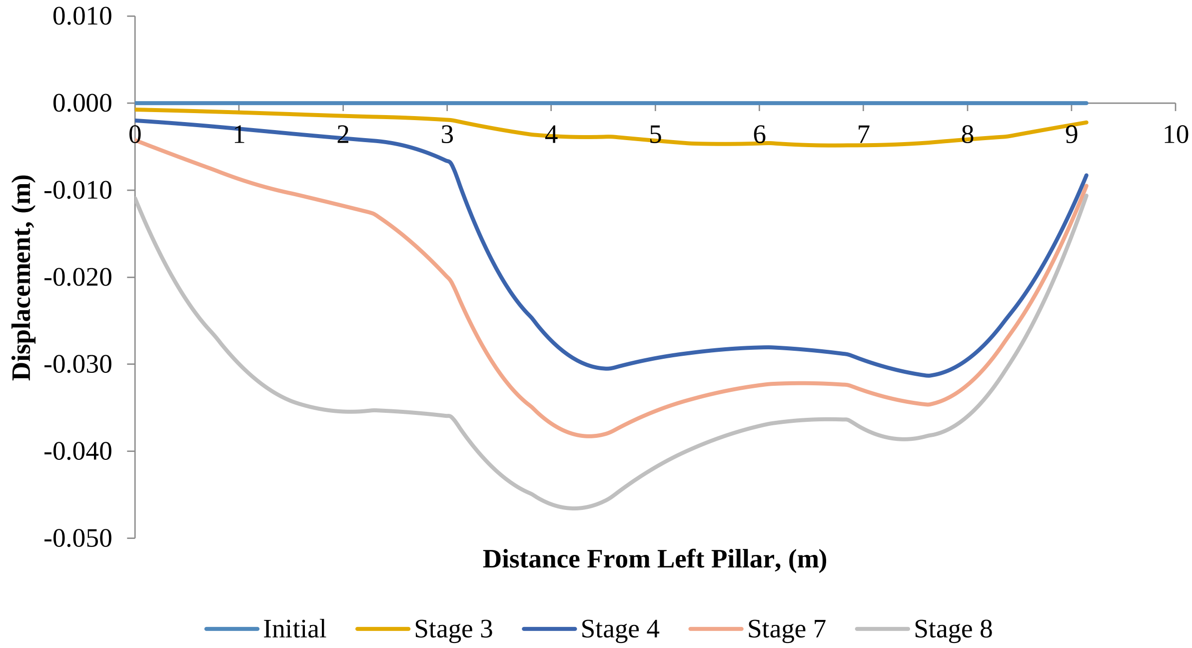


*Figure D.5 - Profile an unbolted tunnel with a 0.46-meter-thick salt beam. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. Overlapping stages are removed for clarity. Extra dips in the mine back begin to appear as the clay seam height is reduced at 4 and 7.5 meters indicating higher likelihood of failure in these areas.*

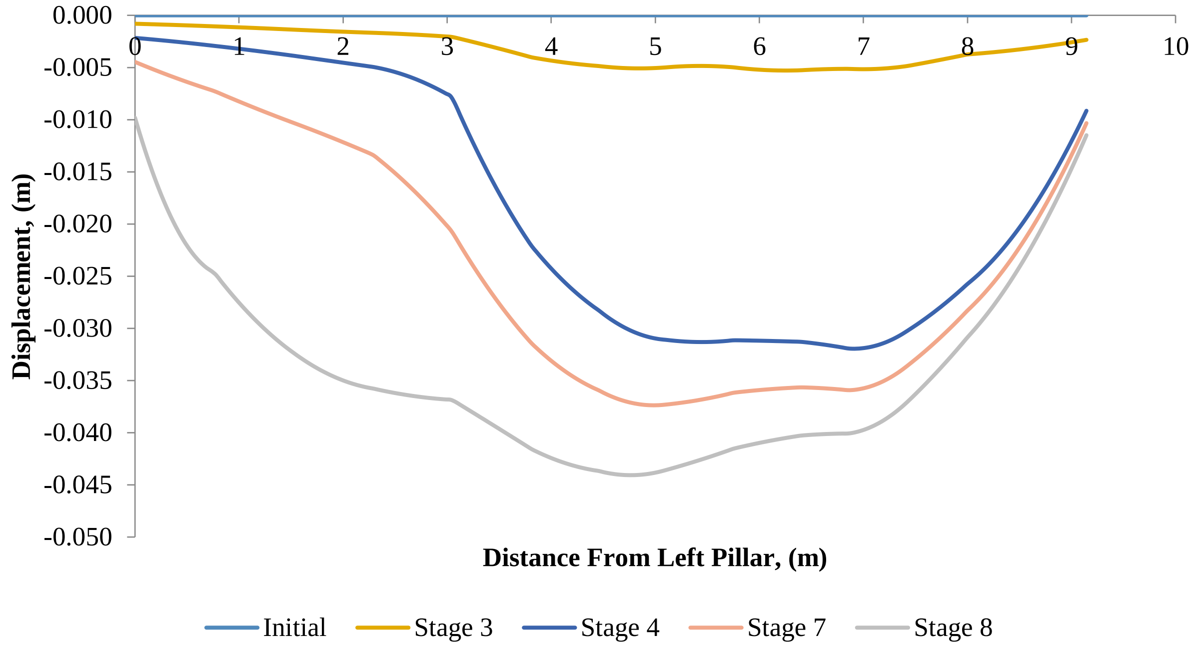


*Figure D.6 - Profile for the proposed partial bolting method, 0.91 meter spacing and a 0.76-meter-thick salt beam. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. Overlapping stages are removed for clarity. With a 0.76-meter thick salt beam, the installation of bolts makes a minimal difference in the displacement profile.*

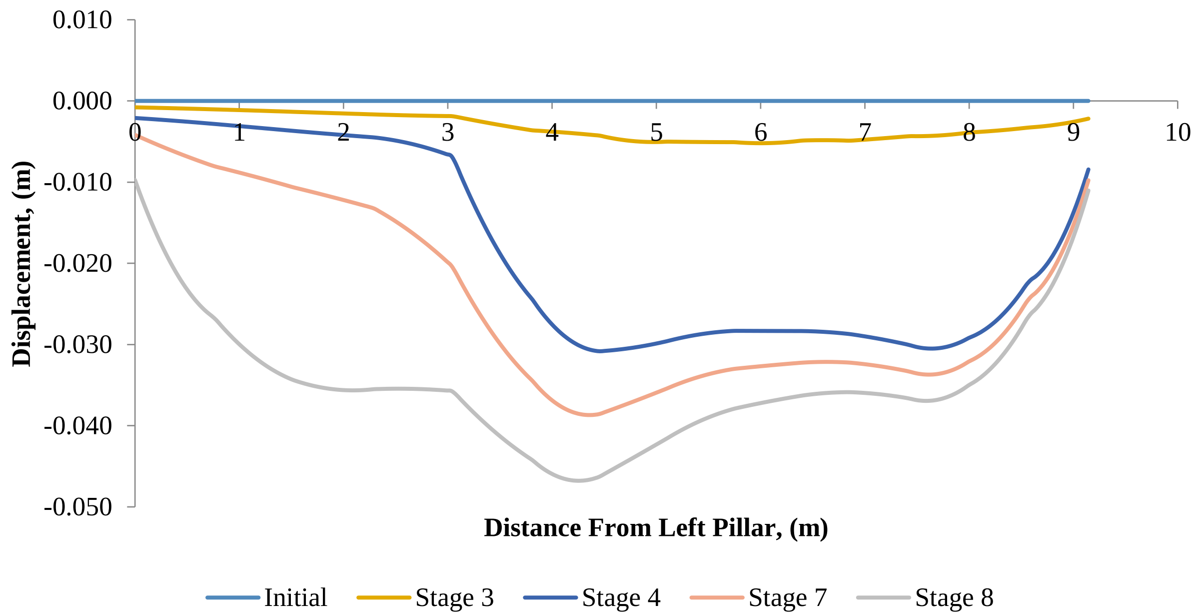


*Figure D.7 - Profile for the proposed partial bolting method, 0.91 meter spacing and a 0.46-meter-thick salt beam. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. Overlapping stages are removed for clarity. In this scenario, the displacement in the area around 7.5 meters is reduced, indicating increased stability in this area.*

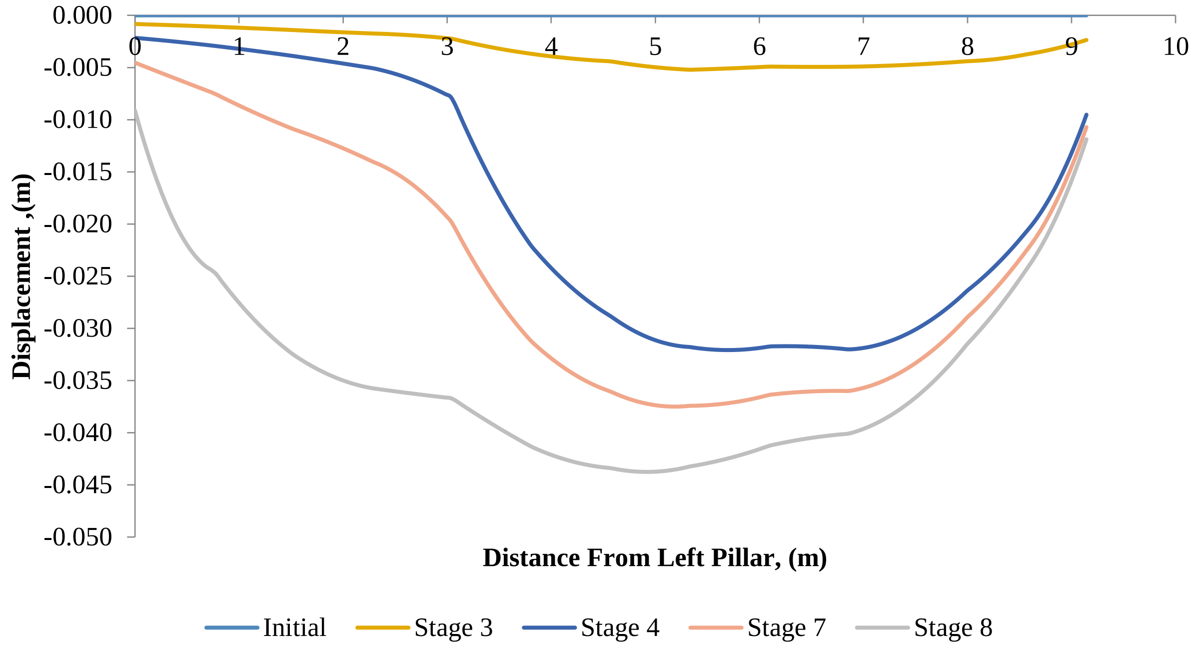


*Figure D.8 - Profile for the proposed partial bolting method, 1.2 meter spacing and a 0.76-meter-thick salt beam. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. Overlapping stages are removed for clarity. With a 0.76-meter clay seam height, the installation of bolts makes a minimal difference in the displacement profile.*

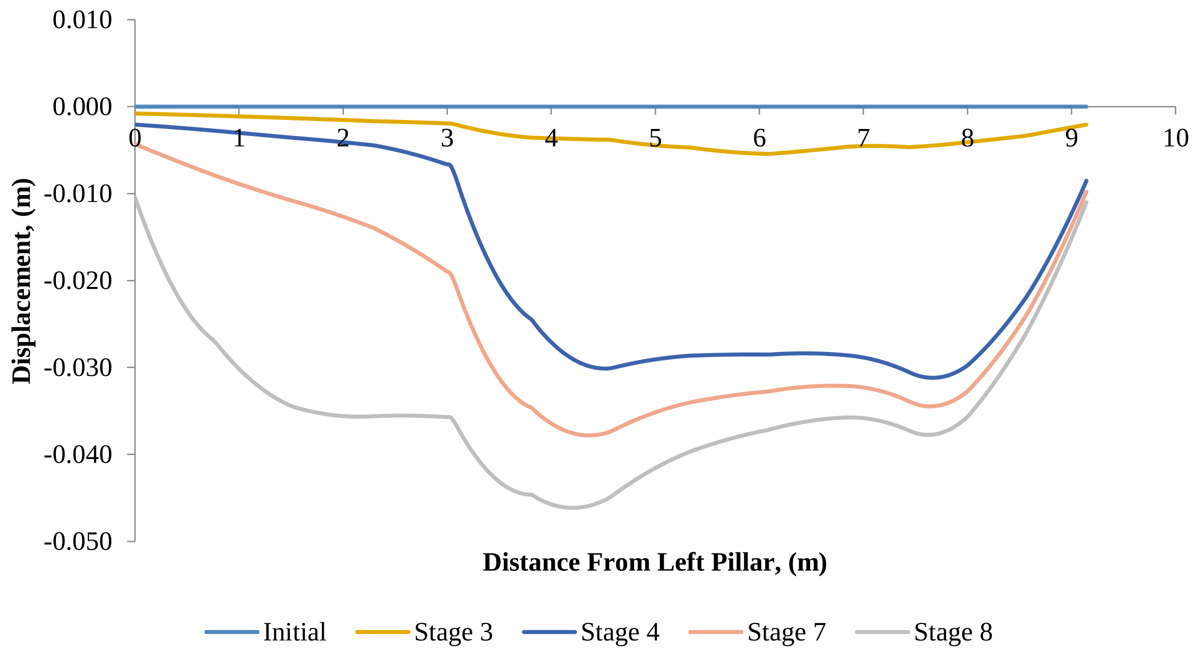


*Figure D.9 - Profile for the proposed partial bolting method, 1.2 meter spacing and a 0.46-meter-thick salt beam. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. Overlapping stages are removed for clarity. Changing the bolt spacing does not make a substantial difference between 0.91 and 1.2 meters.*

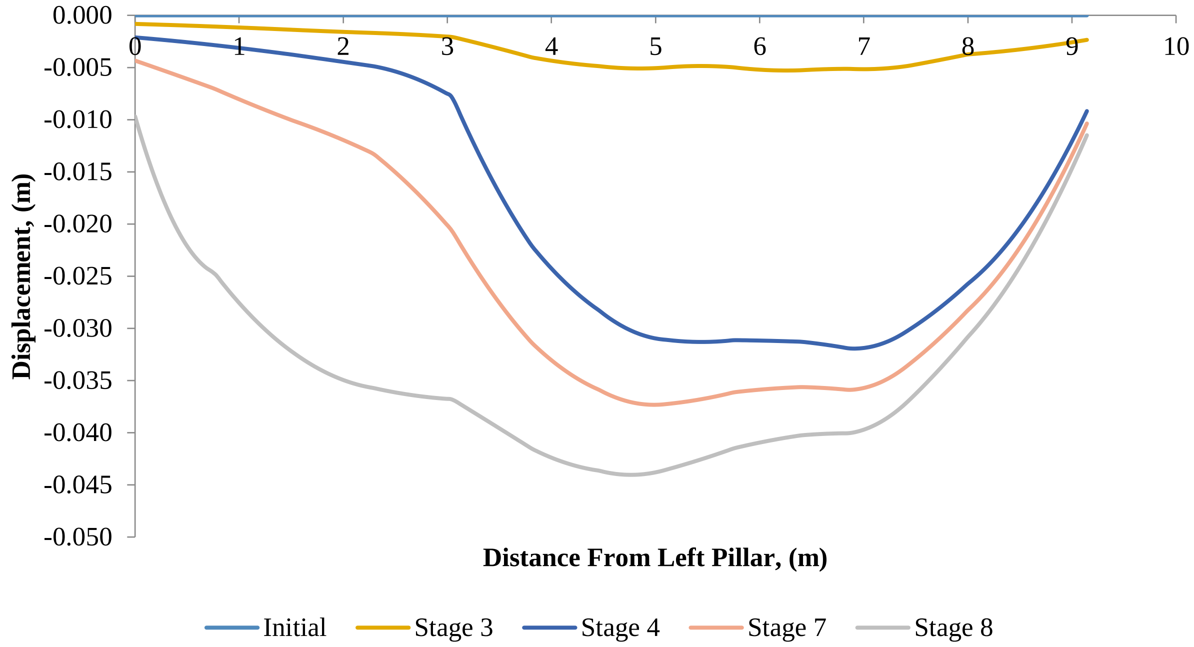


*Figure D.10 - Profile for the proposed full bolting method, 0.91 meter spacing and a 0.76-meter-thick salt beam. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. Overlapping stages are removed for clarity. With a 0.76-meter salt beam, the installation of bolts makes a minimal difference in the displacement profile. Additionally, the installation of bolts across the back of the machine makes a minimal improvement over bolting just the right side of the pass.*

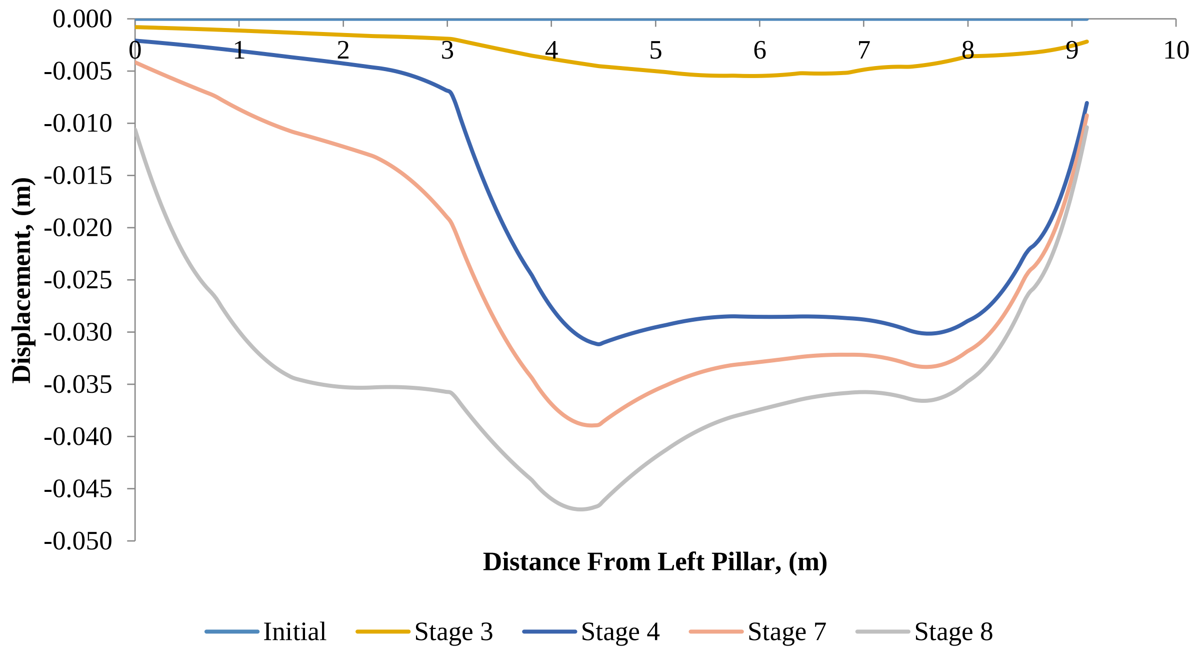


*Figure D.11 - Profile for the proposed full bolting method, 0.91 meter spacing and a 0.46-meter-thick salt beam. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. Overlapping stages are removed for clarity. Additionally, the installation of bolts across the back of the machine makes a minimal improvement over bolting just the right side of the pass.*

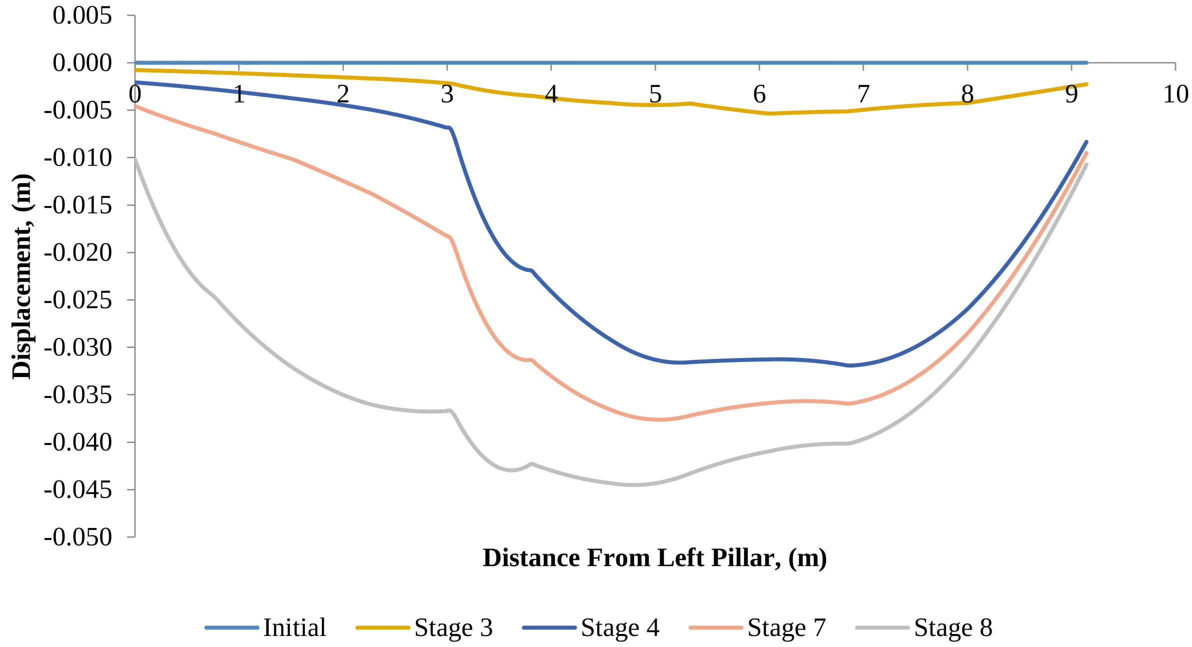


*Figure D.12 - Profile for the proposed full bolting method, 1.2 meter spacing and a 0.76-meter-thick salt beam. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. Overlapping stages are removed for clarity. The increased bolt spacing appears to allow an additional dip in the displacement profile at approximately 3.5 meters.*

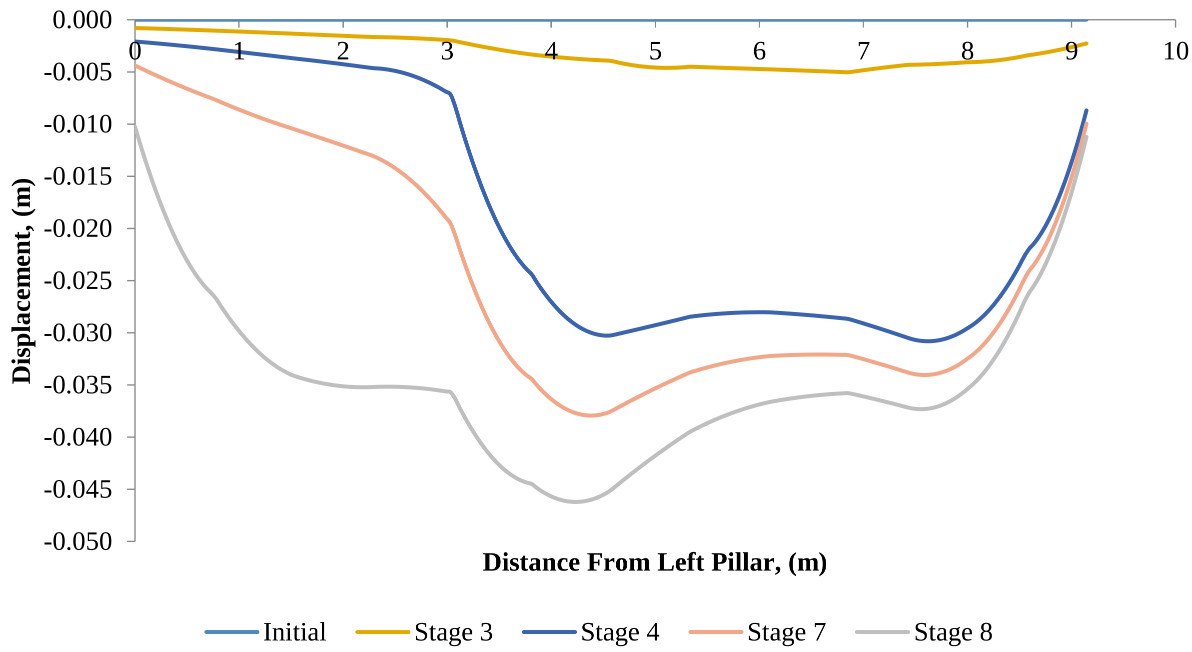


*Figure D.13 - Profile for the proposed full bolting method, 1.2 meter spacing and a 0.46-meter-thick salt beam. Several stages overlap when the stages are far from the line at which the profile is taken, applicable to stages: Initial, Stage 1, and Stage 2; as well as Stage 4, Stage 5, and Stage 6. Overlapping stages are removed for clarity. This profile is again like that shown in Figure D.11 for a 0.91-meter bolt spacing.*